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DEPARTMENT OF THE INTERIOR

HUBERT WORK, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, DIRECTOR

THE ELECTROTHERMIC METALLURGY
OF ZINC

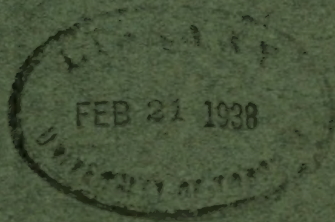
BY

B. M. O'HARRA

THIS BULLETIN REPRESENTS WORK DONE UNDER A COOPERATIVE
AGREEMENT BETWEEN THE BUREAU OF MINES, DEPART-
MENT OF THE INTERIOR, AND THE MISSOURI
SCHOOL OF MINES AND METALLURGY

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THE ELECTROTHERMIC METALLURGY OF ZINC.

By B. M. O'HARRA.

INTRODUCTION.

Zinc smelting is frequently termed a backward art. The term is hardly true, for great progress has been made in recent years in the design and in the thermal efficiency of the retort furnace, in the quality of retorts, in the recovery of zinc, and in the ability to treat more impure and complex ores. The fact remains, however, that the peculiar physical and chemical properties of zinc have delayed such immense advances—large smelting units, high recovery, low-unit costs, and ability to treat low-grade ores—as have been made in the metallurgy of lead and copper.

In current retort-smelting practice, the ores, if carbonates, are calcined to remove carbon dioxide; or, if sulphides, are roasted to remove sulphur. The sulphur must be eliminated as completely as possible, usually to less than 1 per cent. This means that the roast must be continued for a long time at a high temperature, with resulting high cost and small capacity per furnace. The roasted ore is mixed with about 50 per cent of its weight of reducer, which may be fine coke, anthracite, or nonbituminous coal. This mixture is charged into horizontal fire-clay retorts. The retorts used in this country are about 8 inches in internal diameter by 4 feet in length and hold 50 or 60 pounds of ore. The diameter of the retort is limited by the time required for the heat to penetrate to the center of the charge; the length by the strength of the retort, which at the high temperature employed can not support the weight of the charge if the span is more than about 50 inches. The retorts are heated externally by coal, natural gas, or producer gas to a temperature of 1,200° C. or more. The zinc oxide is reduced to zinc, which is volatile at this temperature, and passes into a conical fire-clay condenser attached to one end of the retort, where it condenses to liquid zinc and is removed. Twenty-four hours are required for distillation, and because of the low heat conductivity of the retort walls and of the ore charge itself, the heat efficiency even of the most modern regenerative furnaces is only about 12 per cent. The retorts have a short life, 30 to 60 days, at the high temperature required; zinc

losses are heavy, usually 10 per cent or more; and the proportion of impurities in the retort charge must be carefully controlled to prevent the rapid corrosion of retorts.

For these reasons, much work has been done in an effort to find cheaper and more efficient methods of recovering zinc from its ores. Various methods have been proposed for smelting in the blast furnace. Some of these contemplated the production of zinc vapor and its condensation to liquid; others attempted to produce liquid zinc directly by smelting under pressure. It is now realized that fundamental difficulties render these proposals impracticable.

The electric furnace for smelting offers obvious advantages in the way of the efficient utilization of energy, large units, easy attainment of high temperatures, and the possibility of treating complex ores.

E. H. and A. H. Cowles first suggested a design for an electric furnace for the reduction of zinc ores and patented their process in 1885. Their attention was soon directed to other uses for their furnace and nothing further of importance was done in the electric smelting of zinc until about 1900. Several investigators then started work upon various processes and had more or less success. The chief difficulty they encountered was the impossibility of obtaining more than a small amount of the zinc as liquid metal because most of the zinc vapor condensed as blue powder. Interest in the electrothermic metallurgy of zinc increased, many patents on processes and furnace design were taken out, and the literature on the subject grew, until in 1914 and 1915 the problem seemed to be approaching solution. About this time the hydrometallurgy of zinc, the development of which had been lagging somewhat behind that of the electrothermic metallurgy, sprang into the limelight and a commercial process was soon perfected. This rapid development, perhaps aided by the war premium on the high-grade spelter produced by electrolytic deposition, distracted attention from electric smelting to some extent, though progress continued.

With the perfection and standardization of the electrolytic process has come a better realization of its limitations, especially the necessity for large-scale operation and the difficulty of obtaining high extraction. Because of these limitations the electrothermic metallurgy of zinc still has a field of its own and in fact promises to become a strong competitor of both hydrometallurgy and retort smelting except under conditions highly favorable to the latter methods.

Because of the imperfections of the retort and electrolytic processes and the promise that the electric furnace offers in overcoming these imperfections, the United States Bureau of Mines, in cooperation with the Missouri School of Mines and Metallurgy, has undertaken a study of the electrothermic metallurgy of zinc.

LITERATURE ON ELECTROTHERMIC METALLURGY OF ZINC.

As a first step in this investigation the literature was reviewed thoroughly. This literature is voluminous, but consists mostly of patents and of articles in various periodicals and publications of technical societies. No adequate compilation of the literature exists, so it is difficult for one not having much time at his disposal to acquire a comprehensive knowledge of what has been accomplished. In order that the results of the preliminary study may be available in convenient form for the industry in general, and especially for those who wish to undertake further investigations on their own account, a short review of the work done and the results obtained by previous investigators is presented.

For the purpose of this review, the literature has been drawn on so freely that proper acknowledgment of most of the sources is not possible. The reader is therefore referred for further information to the selected bibliography which is appended, or to the complete "Bibliography on the ElectrotHERMIC Metallurgy of Zinc," Bulletin of the Missouri School of Mines and Metallurgy, volume 6, No. 2.

TYPES OF ELECTRIC FURNACES.

The various furnaces that have been used for smelting zinc ores might be classified as direct-resistance, indirect-resistance, radiating-arc, and buried-arc types; as continuous and intermittent types; or as slagging and dry distillation types. The various processes may be divided into reduction processes, reducing the roasted ore with carbon, as in retort practice; and precipitation processes, in which the raw sulphide ore is treated with copper, iron, or lime and carbon. It has seemed better, rather than follow any of these classifications in describing the different furnaces and methods, to discuss them in the approximate chronological order of their development, beginning with the Cowles furnace, thus giving something of a historical view of the evolution of the whole subject.

ACKNOWLEDGMENTS.

The preparation of this bulletin was undertaken at the suggestion of Dr. C. H. Fulton, director of the Missouri School of Mines and Metallurgy and consulting engineer for the Bureau of Mines, and the writer is greatly indebted to him for his constant advice and helpful criticisms. Especial acknowledgment is made to Mr. F. T. Snyder for information concerning his work in British Columbia and at Chicago; to the Canadian Department of Mines for the use of W. R. Ingalls' unpublished report on the electric zinc smelting experiments of the Canadian Government; and to Mr. O. C. Ralston, of the Bureau of Mines, for information as to costs of power in the western United States.

COWLES BROTHERS' PROCESS.

The first electric furnace for the smelting of zinc ores was that of E. H. and A. H. Cowles, who patented their process in 1885. This furnace consisted of a cylindrical retort *A* (fig. 1) made of silica or other nonconducting material, and similar in form to the ordinary Belgian retort, surrounded by heat-insulating material, *B*. The rear end of the retort cylinder was closed by a carbon plate *C*, which formed the positive electrode, and the outer end by an inverted graphite crucible *D*, which formed the negative electrode and served as a condenser for the zinc vapor. The mouth of the crucible was

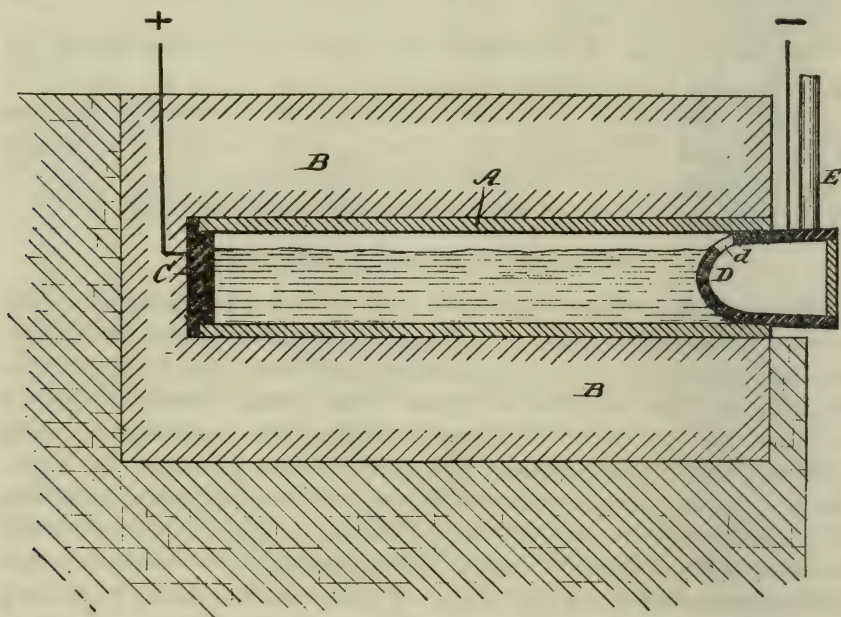


FIGURE 1.—Cowles Brothers' furnace (U. S. Patent 319795): *A*, Cylindrical retort; *B*, heat-insulating material; *C*, carbon plate, forming positive electrode; *D*, inverted graphite crucible, forming negative electrode; *d*, opening in upper side of crucible inside retort; *E*, pipe for escape of gases.

closed with a luting of clay, or otherwise, and the opening *d* in the upper side of the crucible inside the retort formed a passage for the zinc vapor from the retort into the condenser. The pipe *E* permitted the escape of gases from the condenser.

In operation, the roasted zinc ore was mixed with granular carbon, to serve as reduction material and to carry the current, and charged into the retort. The plug *D* was removed for the purpose. The retort was charged nearly full and a small space left along the top. The current was passed through the charge, heating it to distillation temperature; the zinc vapor was then condensed in the graphite

crucible and recovered. A modification was also described which provided more convenient means of charging and discharging the retort.

No further attempt was made to develop this furnace for smelting zinc ores, although the principle was used extensively in the production of aluminum and other difficultly reducible metals.

THE DE LAVAL PROCESS AND ELECTRIC ZINC SMELTING IN SCANDINAVIA.

EARLY HISTORY.

About 1898, C. G. P. de Laval became interested in the electric smelting of zinc ores and a few years later obtained patents on a furnace for this purpose. His first furnaces were of the continuous-smelting arc type shown in Figure 2.

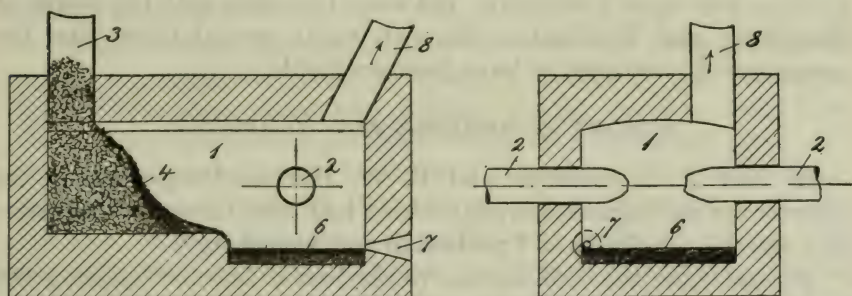


FIGURE 2.—De Laval radiating-arc furnace (U. S. Patent 736611): 1, Furnace chamber; 2, electric arc; 3, charge conduit; 4, surface of the pile of ore; 6, basin; 7, tap hole; 8, outlet for zinc vapor.

The pulverized roasted ore mixed with carbon, or raw sulphide ore mixed with metallic iron, was charged through the conduit 3 at one side of the rectangular furnace, so that it formed a pile sloping toward the source of heat. The surface of the pile 4 was heated by radiation from the electric arc 2, the zinc being reduced and volatilized. The residue melted and flowed down the slope, continually exposing a fresh surface to the action of the heat, and was collected in the basin 6, from which it could be tapped out through the tap hole 7.

The zinc vapor passed through the outlet 8 to a condenser. Later this furnace was modified by feeding the ore in at the back with a screw, thus avoiding the disturbance and evolution of dust from introducing a cold charge on top of the heated surface of the pile. The outlet for the zinc vapor in this improved furnace was placed between the pile of ore mixture and the heating arc to avoid superheating the vapor, which occurred with the original arrangement.

A company was organized in 1903 to exploit de Laval's patents and G. D. L. brand spelter from these works began to appear on the European market in 1904. This was a very pure spelter, containing less than 0.1 per cent total impurities.¹ (This metal was doubtless from the redistillation of scrap zinc or crude spelter, as the metal produced in smelting ores by the de Laval process is apt to be high in lead.) Plants were operated at Trollhattan, Sweden, and Sarpsborg, Norway. Metallurgical results as to condensation and metal recovery from ore were very poor and for several years most of the spelter produced came from the smelting of drosses and scrap zinc; however, by reason of the very cheap power available the company was enabled to continue operation in the face of these metallurgical difficulties and to continue experiments on the smelting of ores. The radiating-arc furnaces were later replaced by a buried-arc or arc-resistance type, but a full description of this type of furnace was never published. Between 1900 and 1911 the works at Sarpsborg and Trollhattan changed hands several times, and the operations do not seem to have been profitable.

REPORT OF MOULDEN AND HARBOARD.

In 1911 J. C. Moulden and F. W. Harboard reported on the process for an English company which had been formed to take over and enlarge the plants at Trollhattan and Sarpsborg.²

At that time the Trollhattan works had a furnace building 300 by 52 feet, equipped with 11 smelting furnaces of the arc-resistance type, 2 refining furnaces, and several arc furnaces of the old type which were not in operation. The works at Sarpsborg had 3 arc-type furnaces and 4 refining ones. The furnaces of the resistance type were of 350 horsepower each and had one large vertical electrode $2\frac{1}{4}$ square feet in cross section, passing through the roof, the other electrode being a carbon block embedded in the bottom of the furnace. The charge was introduced through an opening in the roof, the later furnaces being charged on one side only of the electrode. A furnace was being erected at that time with a continuous side feed which was expected to be an improvement. The capacity of each furnace was about 3 metric tons, and about 2.8 metric tons of ore were actually smelted per 24 hours.

Three-phase current was supplied to the works at 9,640 volts and transformed to a single-phase 100-volt current for use at the furnaces. The ore was charged with suitable flux and reduction material into the furnace, where most of the zinc and some of the lead

¹ Offerhaus, C., Zinc made in the electric furnace: *Electrochem. and Met. Eng.*, vol. 3, 1909, p. 54.

² *Engineering and Mining Journal*, Zinc smelting at Trollhattan; summary of a report by F. W. Harboard: Vol. 93, Feb. 10, 1912, pp. 314-315.

were volatilized and condensed. A large share of this condensed as blue powder and zinc oxide, containing about 54 per cent zinc and 20 per cent lead, which was mixed with fresh ore and recharged. When blue powder was resmelted, a greater proportion of liquid zinc was condensed than when ore alone was smelted.

Most of the lead that escaped volatilization was reduced to metal, carrying most of the silver. Some lead, silver, and zinc with the copper formed a matte, and the rest of the residue fused to a liquid slag. These fused products were tapped from the furnace through a tap hole near the bottom.

Mr. Harboard and his assistants made an extended test on Broken Hill slime containing about 36 per cent zinc, 24 per cent lead, and 30 ounces of silver per ton. Four furnaces worked on ore-powder mixture and three on ore mixture; they took the following charges: Ore-powder mixture—roasted Broken Hill slime, 100 kg.; blue powder, 200 kg.; coke dust, 25 kg.; lime, 5 kg. Ore mixture—Broken Hill slime, 300 kg.; calamine, 10 kg.; coke, 75 kg. At the end of the second day, the ore furnaces were making sufficient powder to supply the powder furnaces (which had been started on powder from the works stock) and no more powder was taken from stock. The furnaces were tapped every four hours for crude zinc and powder, and for slag, matte, and lead about every eight hours. During the early part of the run, the slags contained much zinc and lead, mainly because of imperfect separation of matte. The slags were too basic, from improper proportions of flux and reducing agent. Later on the slags improved, though two furnaces working on the same charge frequently gave entirely different slags, probably because of irregular mixing of the charge outside the furnace.

Improper mechanical facilities for changing electrodes were a serious cause of delay.

During 27.58 days of the experimental run, the seven furnaces smelted 518 metric tons of roasted Broken Hill slime, 19 tons of calamine, and 22.5 tons of blue powder from stock. They produced 160.8 tons of crude zinc and 36 tons of powder and at the end of the run had on hand 13.4 tons more of blue powder than at the beginning, the remainder of the powder having been resmelted. The crude metal obtained from this smelting averaged about 79 per cent zinc, 20 per cent lead, and 0.6 per cent iron. It was afterwards refined, giving 112.4 tons of spelter, 99.9 per cent pure, and 24.7 tons of lead. The lead tapped with the slag was remelted to remove the slag and yielded 41 tons of marketable bullion containing 141 ounces of silver per ton. There was also 17 tons of "leak lead," assaying 27 ounces of silver per ton, and 9 tons of skimmings containing zinc, lead, and silver.

The total input of metals was 204.04 tons of zinc, 128.35 tons of lead, and 15,750 ounces of silver, and the extraction in the form of metals was 130.46 tons of zinc, 94.94 tons of lead, and 7,230 ounces of silver, or a yield of 64 per cent, 73.99 per cent, and 45.9 per cent, respectively. Including the metal content of the powder, these became 73.4 per cent, 79.3 per cent, and 49.5 per cent. A considerable part of the losses of silver and lead were due to absorption by the furnace masonry.

Mr. Harboard considered that side charging would give much better results than charging through the roof. He was of the opinion that the recovery of metals could be much improved, but said that the difficulty of condensing the zinc vapor to liquid zinc was a great drawback.

The loss of zinc in refining 111,794 kg. of crude zinc at Trollhattan was 5.7 per cent. The average results of refining commercial spelter at Sarpsborg were stated to be as follows: Fine zinc produced, 83.9 per cent; powder, 9.8 per cent (corresponding to 8.8 per cent of metals); residues (lead, etc.), 2.2 per cent; charge in furnace, 1.1 per cent; impregnation in brickwork of furnaces and condensers, 0.5 per cent; unaccounted for 3.5 per cent.

A 20-day test at Sarpsborg smelting ore in the old arc-type furnaces gave results similar to those obtained at Trollhattan, but the power consumption was 70 per cent greater. The power used in the experimental run at Trollhattan averaged 2,078 kw. h. per ton of ores melted, including the power used for resmelting the blue powder but not that used for refining crude zinc, the cost of power being 41s. 3d. (\$10) per kilowatt year. The electrode consumption at Trollhattan was 31.51 kg. per metric ton (1,000 kg.) of ore smelted, and at Sarpsborg 40.57 kg. per metric ton of ore.

DEVELOPMENT IN RECENT YEARS.

In 1911 the production of spelter in Sweden and Norway was 6,575 long tons; in 1912, 8,000 tons; and in 1913, 17,000 tons. Part of this came from resmelting of dross and scrap, and part from the smelting of ores.³

A report of the directors of the Hydraulic Power and Smelting Co., issued in 1913,⁴ stated that out of seventeen new 1,000-horsepower furnaces and eight 500-horsepower furnaces planned for Trollhattan, thirteen of the former and all of the latter had been completed. The capacity of the zinc-refining works at Sarpsborg had been increased from 8,000 to 10,000 tons per year and the output

³ Ingalls, W. R., *Electric zinc smelting*: Trans. Am. Electrochem. Soc., vol. 25, 1914, p. 169.

⁴ *Engineering and Mining Journal*, Scandinavian electric zinc smelting: Vol. 96, Nov. 29, 1913, p. 1030.

contracted for at a profit. The smelting of ordinary roasted zinc ores still did not give a profit, but much better results were being obtained.

In 1921, W. R. Ingalls⁵ reported that the plants at Trollhattan and Särpsborg and one or two smaller ones were in operation and that a new plant of an annual capacity of 60,000 metric tons of ore was being built at Glomfjord.

The Trollhattan metallurgists have given up the attempt to obtain liquid spelter from the first smelting operation, but have found that blue powder can be melted economically by heating it to the melting point of zinc and subjecting it to a rubbing action.

The present method of operation is to run the smelting furnace at a very high temperature so that both lead and zinc are completely volatilized. These are condensed as blue powder, which is then melted to a very leady spelter. This metal can be redistilled to yield pure spelter or it may be refined by simple gravity refining to yield an ordinary grade of spelter. Ore-smelting furnaces have been built and successfully operated with a capacity of 12,000 kg. of ore per 24 hours.

DE LAVAL'S CYCLONE FURNACE.

De Laval also obtained patents on a furnace which operated upon an entirely different principle. This furnace is illustrated in Figure 3.

The finely divided ore, flux, and reducer, 2, were fed into a circular furnace, 1, projected tangentially by a gas, such as CO, entering at 5.

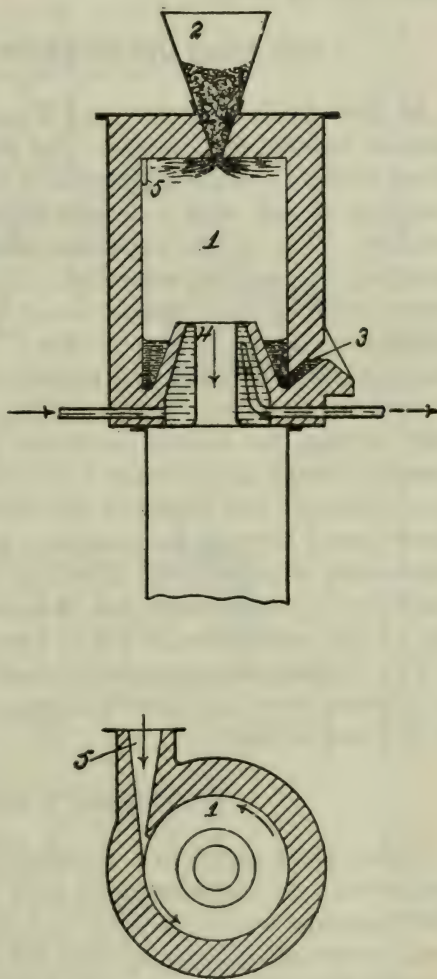


FIGURE 3.—De Laval's "cyclone" furnace (U. S. Patent 852440): 1, Circular furnace; 2, finely divided ore, flux, and reducer; 3, passage through which residue flowed out; 4, exit for zinc vapor and gases; 5, entrance for gas.

⁵ Ingalls, W. R., *Electrometallurgy of zinc*: Trans. Am. Electrochem. Soc., vol. 40, 1921, p. 245.

and were heated to a high temperature. The zinc was reduced and vaporized; the residue was projected against the periphery of the furnace, melted, and ran down the sides to an annular pool at the bottom and flowed out at 3; while the zinc vapor and gases passed out of the furnace by means of the opening 4 in the center. This furnace was tested in England and in Sweden, but a successful operation is not on record.

THE FURNACE OF CASORETTI AND BERTANI.

In 1900 Carlo Casoretti and Francesco Bertani described a continuous furnace for zinc ores (fig. 4) which combined fuel and electrical heating. According to their specifications the fine roasted ore was first mixed with coke and fluxes and briquetted with glucose solution. The bricks were then dried in the regenerator *r* of the smelting furnace and recrushed. The prepared ore was fed into the upper muffle *p* and worked down into the lower muffle *q*, both of which were heated by the coal fire *f*. After being heated in the lower muffle nearly to distillation temperature it was raked with the rake *p'* into the smelting chamber. This was heated by an electric current between the carbon electrodes *f'*. The zinc vapors and gases passed through the passage *v* into the condenser *c*, where the zinc was condensed and collected into suitable receivers. The remaining gases passed through the conduit *v* to the space beneath the muffle *q*, where they were burned. The slag was tapped from the bottom of the electric crucible into the chamber beneath, where it preheated the air for combustion of the CO gas from the condenser.

This furnace was expected to smelt ore with a power consumption of 2 horsepower hours per kilogram of zinc, plus a consumption of 15 per cent of coal.

STANSFIELD'S EXPERIMENTS.

Figure 5 is a sketch of a furnace in which Prof. Alfred Stansfield conducted some experiments in 1900 with a view to smelting Broken Hill lead-zinc ores to recover lead, zinc, and precious metals in the same operation. The charge was fed down the shaft C into the smelting chamber AB, where it was heated by the resistance of the slag bath to the passage of current between the two electrodes. Slag, bullion, and zinc vapor were produced and zinc condensed in the chambers F to K. Some molten zinc was obtained, but the greater part was condensed as blue powder.

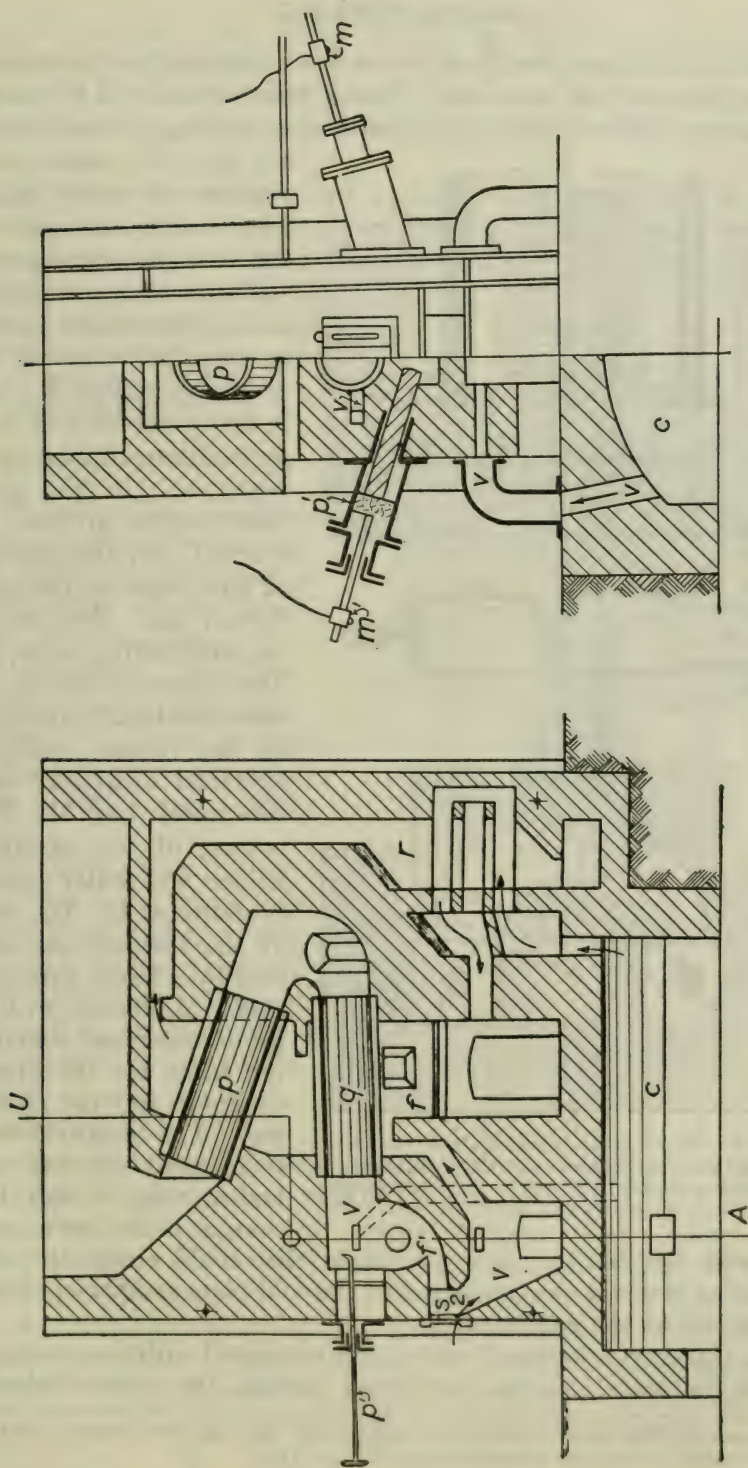


FIGURE 4.—Combined fuel and electric furnace of Casoretti and Bertani (British Patent 472, 1900) : *c*, Condenser; *f*, electrodes; *f*, coal fire; *p*, upper muffle; *p'*, rake; *q*, lower muffle; *r*, regenerator; *v*, passage for zinc vapor and gases; *v'* conduit for gases from condenser.

SALGUES FURNACE.

In 1903, A. Salgues described two or three furnaces for the electrothermic treatment of zinc ores.⁶ One of these is shown in Figure 6. This is one of the earliest adaptations to zinc smelting of the buried-

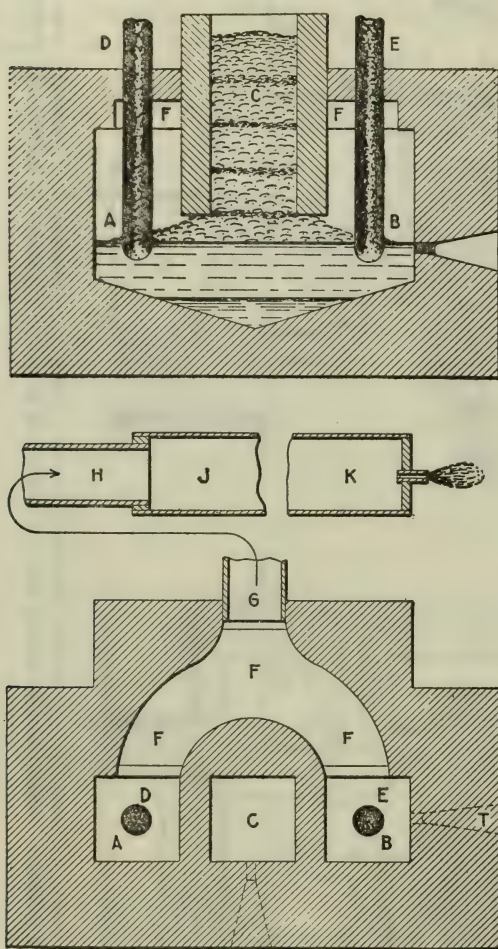


FIGURE 5.—Stansfield zinc furnace (from Stansfield, Alfred, *The electric furnace*, 1914, p. 318): A, B, Smelting chamber; C, shaft; D, E, electrodes; F, G, H, J, K, condensing chambers.

covered with the lids O. Any zinc vapor that could escape through remaining crevices was condensed on the cold plate and the openings were sealed by the condensed zinc.

The charge, consisting of roasted zinc ore mixed with the necessary carbon for reduction, was introduced through the charge holes E

arc type of furnaces, variations of which have been used by so many of the later experimenters.

The smelting chamber, built of fire brick, inside an iron jacket cooled by water sprinklers K, was in two parts—A and B—to facilitate cleaning and repairing. It was provided with a tap hole, D, a flue, C, for the passage of zinc vapor to the condenser, and two charging and poking holes, E. The lower electrode G was embedded in the base of the furnace, making electrical contact with the metal bar H. The bottom of the electrode holder was water cooled as shown at L. The roof of the furnace was covered by a heavy cast-iron plate, M, cooled with a jet of water and containing holes for the upper electrode and for charging. The electrode was surrounded by the asbestos ring N and the charge holes were covered

⁶ Salgues, A., [The electrometallurgy of zinc]: Bull. Soc. Ing. Civ. France, 1903, p. 174; Stansfield, Alfred, *The electric furnace*, 1914, p. 315.

and surrounded the upper adjustable electrode. The furnace operated continuously, producing zinc vapor, lead bullion, matte, and slag. Salgues recognized the necessity for prereduction of the charge if good condensation of liquid spelter was to be obtained.

Salgues's experiments were carried on at Champagne, Ariegne, France, with a modified carbide furnace of 100-kw. capacity. Operating on 40 to 45 per cent zinc ores fed into the furnace cold, he obtained a yield of 5 kg. of zinc per kilowatt day, or 4,355 kw. h. per ton of ore.

JOHNSON PROCESS.

EARLY EXPERIMENTS.

Woolsey McA. Johnson began work on the electric smelting of zinc ores in 1903, while he was metallurgist for the Lanyon Zinc Co. His first efforts followed closely the metallurgy of the retort process, as the Cowles Brothers had done in 1885, but on a larger scale.

His furnace for this purpose (see fig. 7) was a long, box-shaped retort, 1, built of fire brick or other suitable refractory, with an opening, 4, at one end for the passage of the zinc vapor and gases to the condenser. Graphite blocks, 5 and 6, set on opposite sides of the furnace at each end, with their faces flush with the inside of the furnace walls, were connected by means of graphite or metal rods, 7, projecting through the walls to the external electric circuit. Loose, granular coke, 12, or similar material was piled in each end of the chamber between the graphite blocks. The two piles of coke thus formed the electrodes between which the current passed, and the charge itself became the resistor in which the heat was developed. The floor of the furnace was first covered with a protective layer of refractory material, 11, acid or basic, according to the charge to be smelted. Over

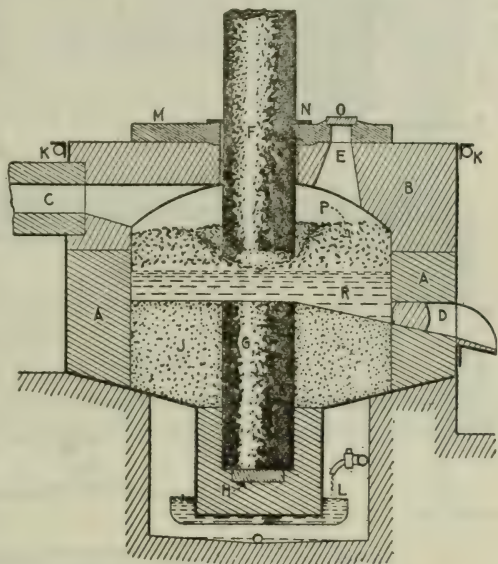


FIGURE 6.—Salgues's zinc furnace (from Stansfield, Alfred, *The electric furnace*, 1914, p. 315): A, B, Walls of smelting chamber; C, flue; D, tap hole; E, charging and poking holes; F, upper electrode; G, lower electrode; H, metal bar; K, water sprinklers; L, water for cooling bottom of electrode holder; M, cast-iron plate; N, asbestos ring; O, lids; P, charge; R, molten metal and slag.

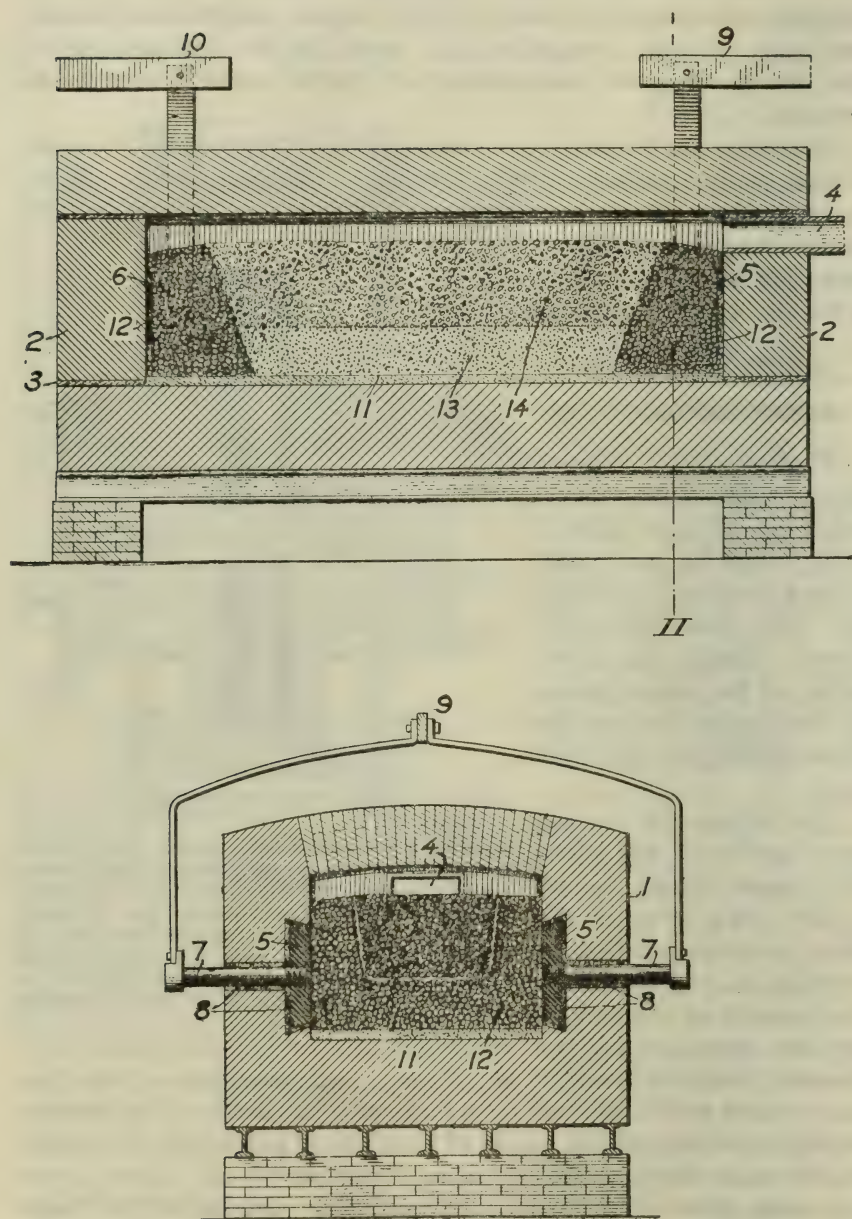


FIGURE 7.—Early furnace of W. McA. Johnson (U. S. Patent 814050): 1, Retort; 2, furnace walls; 4, opening for passage of zinc vapor and gases to condenser; 5, 6, graphite blocks; 7, graphite rods; 8, insulation; 9, 10, electric connection; 11, refractory material; 12, coke; 13, mixture of high-grade ore and coke; 14, mixture of low-grade ore and coke.

this refractory material and extending upward against the side walls was placed a portion, 13, of the charge, consisting of high-grade roasted ore mixed with sufficient high-resistance coke to act as reducer and to carry the electric current. The space above and within this high-grade charge was filled with a mixture of low-grade ore and coke, 14. In this way the high-grade ore protected the furnace walls from the corrosive action of the low-grade ore. As described in the patent, it was necessary to remove one or both end walls to rake out the residue and introduce a new charge. These walls were made of single blocks, or doors, which could be easily removed and could be luted in when the furnace was in operation. Other openings could be arranged if desired.

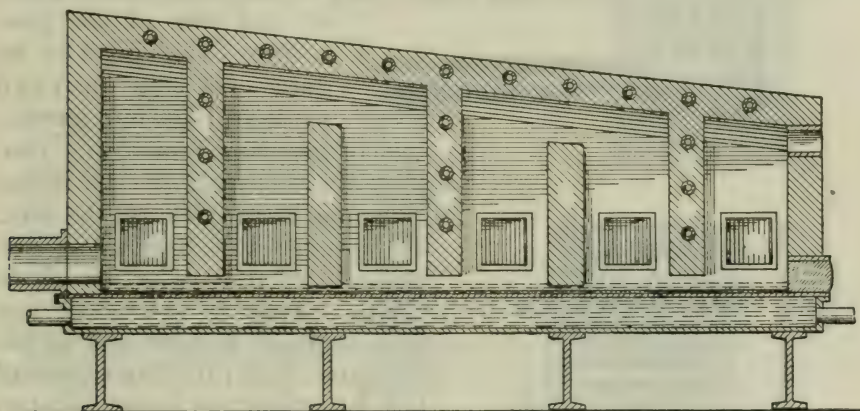


FIGURE 8.—An early Johnson condenser (U. S. Patent 851520).

CONDENSATION DIFFICULTIES.

Johnson had the usual difficulties with condensation and much of his effort in the following few years was turned toward solving these difficulties. He tried condensers of several designs and obtained patents on them. Two are shown in Figures 8 and 9. In both of these arrangement was made to conduct the vapor through a tortuous passage with artificially cooled walls, and to bring it into contact with a body of molten zinc. In the condenser shown in Figure 9 there was a continual shower of molten zinc overflowing from the receptacles A. This shower was kept up, when necessary, by pumping molten zinc from the sump to the top of the condenser. In practice this was found to be less satisfactory than the stationary bath of zinc.

PREREDUCTION OF THE CHARGE.

The desirability of a prereduction of the charge, both as a help in obtaining a proper condensation of zinc and as an economy of electrical power, was early realized.

The earlier method of carrying this out is described in United States patent 868345.⁷ The ore, after suitable crushing, was roasted at a low temperature, about 850° to 900° C., in any suitable roaster. It was not necessary to reduce the sulphur below about 3 per cent, so that if the roasting was done in a muffle-type roaster a rich SO₂ gas, very desirable for sulphuric acid making, could be produced.

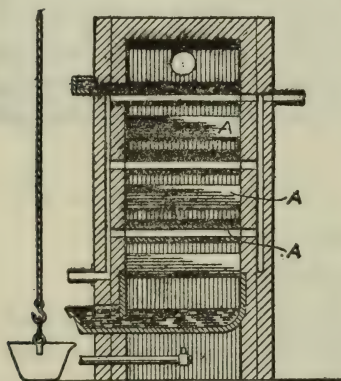
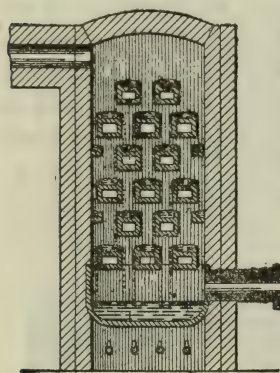


FIGURE 9.—Another early Johnson condenser (U. S. Patent 902534) : A, Receptacles for molten zinc.

The roasted ore was mixed with a suitable proportion of carbon, 35 per cent, for example, and heated to a temperature of 850° to 950° C. This reduced iron oxide to sponge iron, but left zinc oxide unaffected. If desirable, natural gas or some other reducing agent could be substituted for carbon in this step of the procedure. This partly reduced ore was charged while still hot into an electric furnace and heated to a temperature sufficient for the reduction and distillation of the zinc.

The sponge iron formed in the preliminary part reduction of the charge served to decompose any zinc sulphide present and to reduce zinc oxide; the importance of this latter reaction depended upon the amount of iron in the ore.

By following this procedure as outlined, the CO₂ and CO produced in the reduction of iron are eliminated in the preliminary stage; consequently in the final stage a rich zinc vapor is formed which is more easily condensed to liquid metal.

TYPE OF FURNACE USED IN LATER EXPERIMENTS.

While the condensation problem was being worked out, various types of furnace were experimented with, until that used in the later larger scale tests was developed. This final type, as first described, is illustrated in Figure 10. It was of the continuous-smelting, buried-arc class, similar to the Scandinavian and Salgues furnaces, producing spelter, bullion, matte, and slag in the same operation.

⁷ Johnson, W. McA., Process of treating ferruginous blende, U. S. Patent 868345. Oct. 15, 1907.

The electrodes 3, projecting vertically through the roof of the furnace 1 were connected to one terminal of the external electric circuit while the electrode 5, embedded in the hearth of the furnace, was connected to the opposite terminal. The charge was introduced through the hopper 7. Metal and matte were removed through the tap hole 9 and slag through the slag tap 4. The zinc vapor and gases passed by means of the conduit 12 to the column of coke 13 which was so arranged that it could be heated to incandescence, preferably by electrical means, and thence to the condenser 21.

In the operation of this furnace the ore was charged into the furnace with only sufficient carbon to reduce the valuable metallic constituents, leaving no excess. Proper fluxes were added to form a slag fusible at $1,100^{\circ}$ to $1,175^{\circ}$ C. Under these conditions a con-

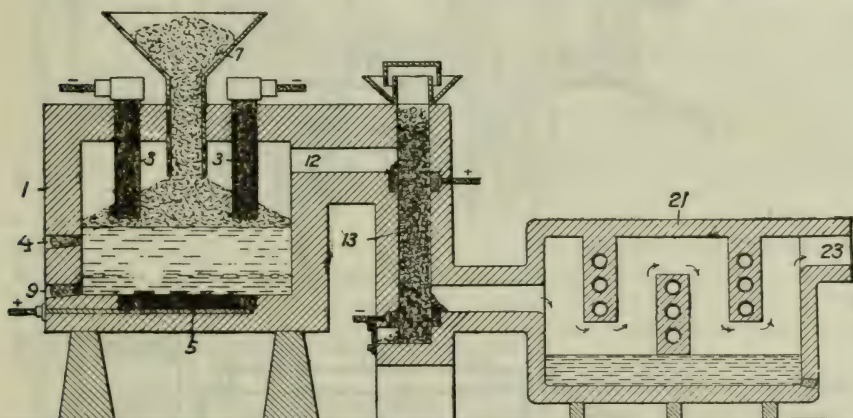


FIGURE 10.—Johnson continuous zinc furnace (U. S. Patent Reissue 13208) : 1, Furnace ; 3, electrodes ; 4, slag tap ; 5, electrode ; 7, hopper ; 9, tap hole ; 12, passage to condenser ; 13, coke ; 21, condenser.

siderable proportion of CO_2 was formed in the smelting furnace, and the hot-coke column was introduced to reduce the CO_2 to CO and prevent oxidation of the zinc vapor.

Figure 11 shows the construction of a later form of this furnace. It was found possible at a later stage of the experiments to do away with the coke column.

EFFECTS OF SULPHUR ON THE CONDENSATION OF ZINC.

In this later work, Johnson found that sulphur in the electric furnace, either as free sulphur, carbon disulphide, sulphur dioxide, or sulphur trioxide, tended to sulphidize the zinc vapors and form blue powder. He then proposed to modify his procedure to avoid this trouble.⁸ He so conducted the roasting operation at low tem-

⁸ Johnson, W. McA., Electric reduction of zinc ores, U. S. Patent 1150271, Aug. 17, 1915.

perature as to reduce the sulphur to the desired minimum and to decompose all sulphates other than those of lead, calcium, and barium. The roasted ore was then mixed with reducing agents, say 15 to 20 per cent coal, and fluxes, and treated in the preheater in such a way as

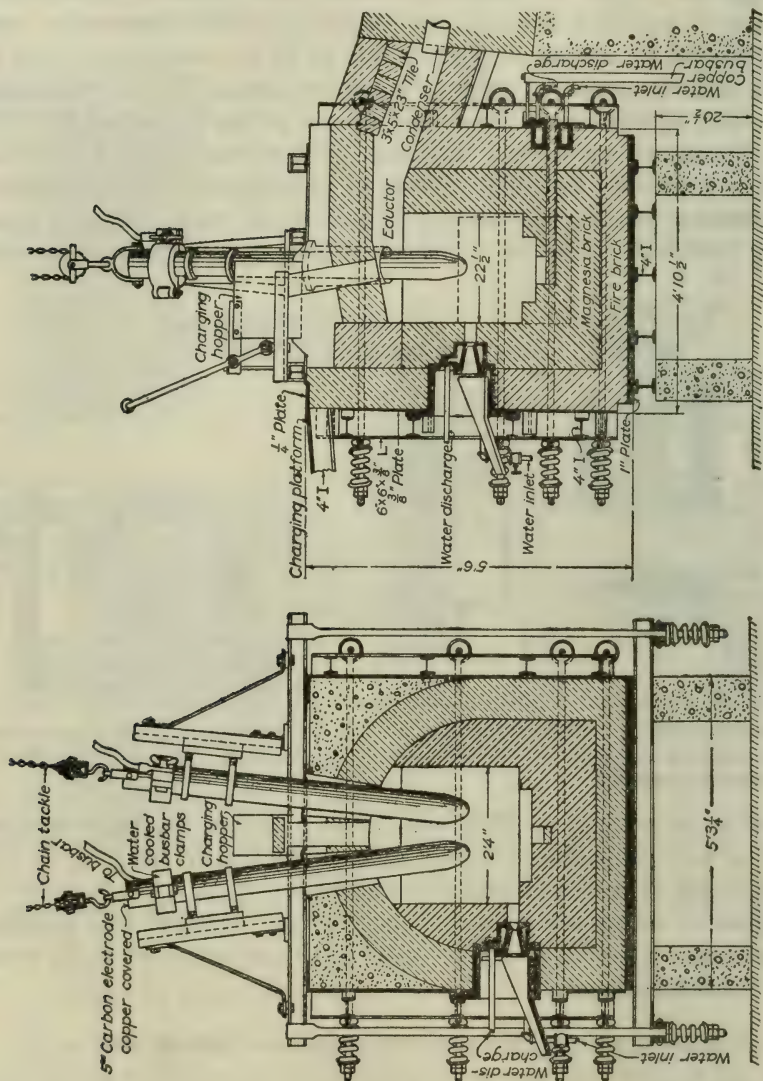


FIGURE 11.—A later form of the Johnson continuous furnace (from Engineering and Mining Journal, vol. 96, 1913, p. 965).

to reduce the sulphates present to sulphides and to reduce iron oxides and copper oxides to metal. Any muffle-type roasting furnace could be used as a preheater. The charge to the electric furnace was so proportioned that the percentage of iron and copper present should be from five to seven times that of sulphur. These metals decomposed any sulphides present in the charge, forming copper and iron

subsulphides. Because of the excess of metallic copper and iron present, no sulphur vapors could be set free and sulphidization of the zinc vapor was prevented.

RESULTS OF TESTS OF JOHNSON'S PROCESS.

By 1912, Johnson's process had begun to give very promising results. During 1912, 1913, and 1914 he made continuous runs in furnaces producing several hundred pounds of zinc per day, and published the results of several of these, which were more or less satisfactory. In 1914, he read a paper before the Canadian Mining Institute, giving a description of the process and the furnace as developed at that time, and the metallurgical results of a typical furnace run. The following description is taken from that paper.⁹

The process, as applied to zinciferous copper-lead ores, is briefly as follows:

(1) Rough roasting, to expel a large part of the sulphur until it does not exceed, say, from 4 per cent to 6 per cent. This roasting is an ordinary operation and can be done in general practice at a cost of less than \$1 per ton. If the original ore or concentrate is high enough in sulphur, it can be roasted by sulphuric acid manufacturés at little or no charge to the metallurgist and the cost of this preliminary treatment would then be reduced to that of freight and handling only.

(2) Preheating and prereducing the ore with admixture of soft coal, so as to reduce much of the iron oxide to metallic iron (sponge or pyrophoric iron), which is necessary in the subsequent treatment in the electric furnace. More than half of the heat required for the chemical reactions is supplied very cheaply in this preheater.

(3) Final reduction to metals in the continuous zinc furnace to decompose the sulphides of zinc and lead, forming (a) metallic zinc which is volatilized and collected in the condenser, (b) copper matte, and (c) lead bullion which are collected in the bottom of the furnace, and (d) molten slag which floats on the top of the matte, is the most important step in the process.

The present Johnson continuous electric zinc furnace, as installed at Hartford, has an exterior form of a cube about 5 feet on the edge, resting on a heavy iron plate supported by four piers. The bricks of the furnace are bound together by 4-inch I beams and tie-rods having heavy steel springs at the ends to compensate for expansion and contraction. The outer construction of fire brick surrounds the interior hearth or crucible of magnesite brick. This crucible, in which the smelting reactions occur, has interior dimensions of 35 inches by 40 inches by 30 inches in height. At the bottom of the crucible is a water-cooled tap hole for the molten matte and lead bullion, and at a proper distance from the bottom is another water-cooled tap hole for the removal of the molten slag. Four copper-covered carbon electrodes 5 inches in diameter are attached to the bus bars by water-cooled clamps. These electrodes, supported by chain tackles, project through the arched roof of the furnace into the crucible. Hoppers for the introduction of ore, flux, and reducing material are situated at the top of the furnace near the electrodes. At one side near

⁹ Johnson, W. McA., The commercial aspect of electric zinc-lead smelting: Trans. Canadian Min. Inst., vol. 17, 1914, pp. 107-129.

the top is a brick-lined flue to carry the vaporized zinc and furnace gases to the condenser in which the zinc is collected in molten metallic form, and the escaping gas, chiefly carbon monoxide, is burned in a steady flame as it escapes through small orifices into the air. * * *

The ores to be treated in the electric furnace are crushed to proper size, roasted to proper content of sulphur, and blended to give a mixture of the following range of composition: Zinc, 15 to 40 per cent; lead, 5 to 30 per cent; copper, 1 to 5 per cent; iron, 10 to 20 per cent; sulphur, 3 to 7 per cent; silica, 10 to 30 per cent; alumina, 3 to 5 per cent; and lime, 3 to 7 per cent. A type mixture would contain: Zinc, 30 per cent; lead, 10 per cent; copper, 2 per cent; iron, 15 per cent; sulphur, 4 per cent; silica, 15 per cent; and lime, 6 per cent. This blended material is then mixed with about 15 per cent of soft coal and carefully charged into a muffle preheater of the Hasenclever type, fired with soft coal, and carefully deoxidized at a final temperature of from 850° C. to 1,040° C., which is just below the reduction temperature of zinc oxide. The deoxidized material is then fed continuously into the electric furnace, which is of the buried-arc type. The electrodes are submerged in the slag and the charge. Care must be taken that there is no local heating, called "hot spots," or local cooling, called "cold spots" or "freezing," of the slag in the fusion zone, which should have a temperature of from 1,225° C. to 1,250° C.

In the preheating operation the following chemical reactions occur:

- (1) $\text{CaSO}_4 + 2\text{C} + \text{H}_2 = \text{CaS} + \text{CO} + \text{CO}_2 + \text{H}_2\text{O}.$
- (2) $\text{BaSO}_4 + 2\text{C} + \text{H}_2 = \text{BaS} + \text{CO} + \text{CO}_2 + \text{H}_2\text{O}.$
- (3) $\text{ZnSO}_4 = \text{ZnO} + \text{SO}_3 = \text{ZnO} + \text{SO}_2 + \text{O}.$
- (4) $2\text{Fe}_2\text{O}_3 + 3\text{C} + 2\text{H}_2 = 4\text{Fe} + 2\text{CO} + \text{CO}_2 + 2\text{H}_2\text{O}.$
- (5) $4\text{CuO} + 2\text{C} + \text{H}_2 = 4\text{Cu} + \text{CO} + \text{CO}_2 + \text{H}_2\text{O}.$

In the above reactions, the carbon is derived from solid carbon, carbon monoxide, and hydrocarbons. The hydrogen comes from the hydrocarbons. The relative percentages of CO and CO₂ in the furnace gases will depend on the physicochemical conditions of the charge. There are also other chemical reactions which need not be given here.

From 90 to 99 per cent of the Fe₂O₃ is reduced to the metallic state. ,

The charge, after the desired reactions have taken place in the preheater, is fed through the hoppers of the electrical furnace direct to the smelting zone in which the following chemical reactions occur, under the influence of heat generated by the passage of the electric current:

	Temperature at which reactions begin, °C.
6. $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$	1, 040
7. $\text{ZnS} + \text{Fe} = \text{Zn} + \text{FeS}$	1, 175
8. $\text{ZnS} + 2\text{Cu} = \text{Zn} + \text{Cu}_2\text{S}$	1, 150
9. $\text{CO}_2 + \text{Fe} = \text{FeO} + \text{CO}$	1, 000
10. $\text{ZnO} + \text{Fe} = \text{FeO} + \text{Zn}$	1, 200
11. $3\text{Fe} + \text{SO}_2 = \text{FeS} + 2\text{FeO}$	1, 200
12. $2\text{ZnS} + \text{C} = 2\text{Zn} + \text{CS}_2$	1, 350
13. $2\text{Fe} + \text{CS}_2 = 2\text{FeS} + \text{C}$	1, 100
14. $\text{ZnO} + \text{CO} = \text{Zn} + \text{CO}_2$	900

As a result of these reactions, the sulphur combines first with the reduced copper and then with reduced iron to form copper matte, which collects some silver and gold and settles to the crucible. Similarly the reduced lead, in molten metallic form, collects the rest of the silver and gold, forming a base bullion, which settles in the crucible and collects below the molten matte. The silica, lime, iron oxide (and alumina and magnesia if present) combine to form the slag, which being specifically lighter, floats on top of the matte.

The zinc, which is reduced to the metallic form, is immediately vaporized at the temperature of the furnace and passes to the condenser at the side, where it is condensed and collected in a molten condition. The slag is deoxidized and desulphurized and the matte is made so metallic that no sulphur vapor is given off at the temperature of the fusion zone. Under this condition the gas from the furnace is so pure that when it reaches the condenser the zinc vapor changes directly from the gaseous condition to the liquid condition, and molten zinc is formed with little or no blue powder.

The condenser consists of elongated iron vessels lined with fire brick having movable covers to facilitate cleaning, and capable of being heated or cooled. If sufficient service be provided and enough time allowed, the zinc vapor will condense satisfactorily to the molten metal. The uniform manner in which the chemical reactions occur in the smelting furnace is shown by the steadiness of the flame of burning carbon monoxide as it escapes into the outer air from the prolongs of the condensing vessels. This flame has the appearance of that of a Bunsen burner in a laboratory.

Johnson emphasized the fact that the field of his furnace lay especially in the treatment of complex ores of zinc, lead, copper, and precious metals, which could not be treated by the retort process, or could only be treated at a higher cost than the purer high-grade ores. In the electric furnace smelting of preheated ore, the chief consumption of power was in the reduction of zinc oxide. The lower-grade ores could, therefore, be treated more cheaply per ton of ore than high-grade ores; and the lead, copper, and precious metals could be recovered at the same time as the zinc.

Table 1¹⁰ gives the metallurgical balance sheet of run No. 1 in Johnson's electric furnace No. 26, January 3 to February 13, 1914.

TABLE 1.—*Metallurgical balance sheet of run No. 1, in Johnson electric furnace No. 26, January 3 to February 13, 1914.^a*

(This balance sheet is from the mixing floor to the electric furnace products.)

ZINC.			
Charge chiefly.	Quantity, pounds.	Per cent.	Metal content, pounds.
Colorado ore.....	27,847	31.6	8,835
N. H. ore.....	47,248	42.4	11,526
Y. P. slimes (raw).....	5,430	33.7	1,831
			22,192

^a The energy consumption per ton of charge was 1,490 kw. h. Average voltage, 25; average amperes, 2,800.

Theoretical energy consumption with efficient prereducing, per short ton:

(Zn. 29 per cent) calcined zinc ore 264 kw. h.

(Zn. 44 per cent) calcined zinc ore 565 kw. h.

(Zn. 70 per cent) calcined zinc ore 900 kw. h.

Electrode carbon consumed in 41 days, 185 pounds, or 4.5 pounds per day; per ton of ore treated, 6.05 pounds.

¹⁰ Johnson, W. McA., The commercial aspect of electric zinc-lead smelting: Trans. Canadian Min. Inst., vol. 17, 1914, pp. 120-122.

Accounted for.	Quantity, pounds.	Per cent.	Metal content, pounds.	Distribu- tion of metal, per cent.
Spelter	17,245	98.0	16,900	76.3
Flue dust	2,594	68.2	1,769	8.0
Matte	3,478	9.1	316	1.4
Lead dross	177	4.0	7	---
Slag	45,850	6.9	3,156	14.2
			22,148	99.8
In furnace bottom not yet recovered	---	---	44	0.2
			22,192	100.0
				Per cent.
Spelter recovery ^b				77.9
Zinc recovered in condenser				8.2
Total zinc recovery				86.1
Condensation factor ^b				77.87

LEAD.

Charge chiefly.	Quantity, pounds.	Per cent.	Metal content, pounds.
Colorado ore-----	27,847	12.75	3,556
N. H. ore-----	27,248	6.13	1,671
Y. P. slimes (raw)-----	5,430	8.63	467
			5,694

Accounted for.	Quantity, pounds.	Ounces per ton.	Metal content.	Distribu- tion of metal, per cent.
Bullion -----	-----	-----	2,437.0	42.8
Matte -----	3,478	9.07	315.3	5.5
Lead dross-----	177	60.0	106.2	1.9
Slag -----	45,850	0.153	70.4	1.2
Spelter -----	17,245	2.03	351.5	6.1
Flue dust-----	2,562	14.5	371.0	6.5
In furnace bottom not yet re- covered -----	-----	-----	2,042.6	36.0
			5,694.0	100.0

COPPER.

Charge chiefly.	Quantity, pounds.	Per cent.	Metal content, pounds.
Colorado ore	27,847	1.69	470.0
N. H. ore	27,248	0.64	174.4
Y. P. slimes (raw)	5,430	0.57	30.3
			674.7

^b "Spelter recovery" is calculated in the usual commercial way—i. e., on the basis of 100 per cent of zinc in the spelter. The "condensation factor" is similarly calculated.

Accounted for.	Quantity, pounds.	Per cent.	Metal content, pounds.	Distribu- tion of metal, per cent.
Matte -----	3,478	12.62	439.0	65.2
Slag -----	45,850	0.239	109.3	16.2
Spelter -----	17,245	0.05	8.6	1.3
Flue dust -----	2,594	0.07	1.8	0.3
In furnace bottom not yet re- covered -----	-----	-----	116.0	17.0
			674.7	100.0

SILVER.

Charge chiefly.	Quantity, pounds.	Ounces per ton.	Metal content, ounces.
Colorado (brown ore) -----	13,288	22.0	146.0
Colorado (black ore) -----	12,664	9.0	61.5
N. H. No. 1 -----	16,427	6.2	51.0
N. H. No. 2 -----	4,054	2.6	5.3
Slimes (reduced) -----	1,831	4.69	4.3
Old mix -----	6,290	5.35	16.8
			284.9

Accounted for.	Quantity, pounds.	Per cent.	Metal content, pounds.	Distribu- tion of metal, per cent.
Bullion -----	2,437	116.5	142.9	50.7
Matte -----	3,478	21.8	37.9	13.5
Lead dross -----	177	87.0	7.7	2.7
Spelter -----	17,245	0.45	3.9	1.4
Flue dust -----	2,594	0.45	0.6	0.3
Slag -----	45,850	0.78	17.9	6.4
In furnace bottom not yet recovered -----	-----	-----	74.0	30.0
			284.9	100.0

GOLD.

Charge chiefly.	Quantity, pounds.	Ounces per ton.	Metal content.
Colorado (brown ore) -----	13,288	0.04	0.268
Colorado (black ore) -----	12,664	0.44	3.008
N. H. No. 1 -----	16,427	-----	-----
N. H. No. 2 -----	4,054	-----	-----
Y. P. Slimes (reduced) -----	1,831	0.05	0.046
Old mix -----	6,290	-----	-----
			3.322

Accounted for.	Quantity, pounds.	Ounces per ton.	Metal content.	Distribu- tion of metal, per cent.
Bullion -----	2,437	1.07	1.359	45.0
Matte -----	3,478	0.099	0.173	5.8
Lead dross -----	177	3.0	0.266	8.9
Spelter -----	17,245	-----	-----	-----
Flue dust -----	2,594	-----	-----	-----
Slag -----	45,850	-----	-----	-----
In furnace bottom not yet recovered -----	-----	-----	1.520	40.3
			3,318	100.0

LATER DEVELOPMENTS.

After Johnson had convinced himself that the various stages of his procedure were properly correlated he proposed various methods of combining the furnaces used for the different steps into a self-contained unit to carry out the complete process cheaply and efficiently. One of these combinations is shown in Figure 12, in which *A* is a fuel-fired muffle furnace, *B* the electric smelting furnace, and *C* the zinc condenser. The previously roasted ore, mixed with reducer and flux, is fed in at the top of the muffle furnace and rabbled from

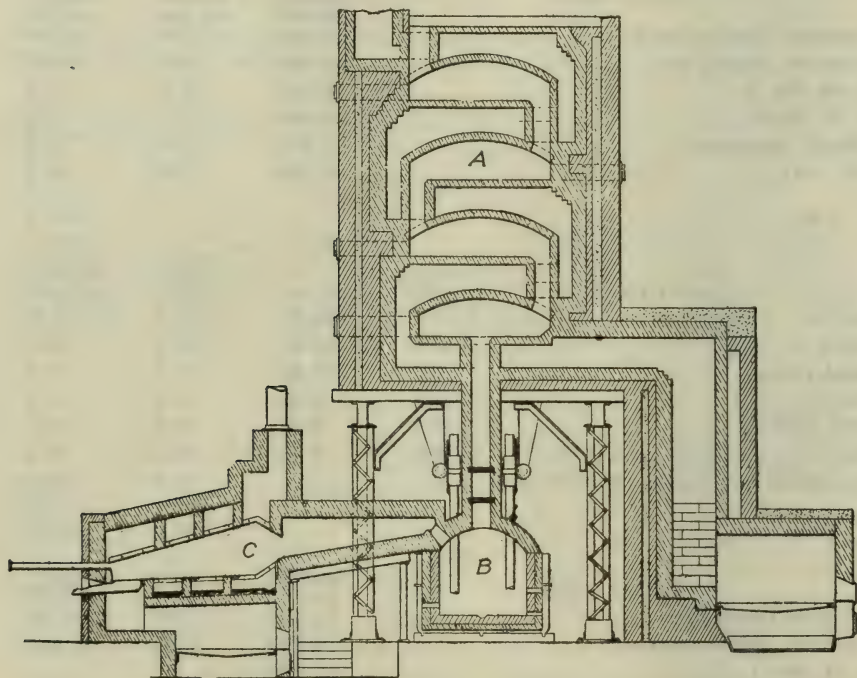


FIGURE 12.—Johnson continuous zinc smelter (U. S. Patent 1244504) : *A*, Fuel-fired muffle furnace; *B*, electric smelting furnace; *C*, zinc condenser.

one hearth to another until the prereduction is complete. It then goes from the lower hearth through a chute directly into the electric furnace below, in which it is smelted and the zinc vapor led into the condenser as shown.

In 1914 Johnson was reported to be planning a 10-ton plant at Keokuk,¹¹ and in 1917 the major rights in his patents were sold to the American Smelting & Refining Co.,¹² who were to try them out at one of their western plants, but no results of either of these trials have been published.

¹¹ Bartlett, R. L., *Zinc: Mineral Industry*, vol. 23, 1914, p. 798.

¹² *Metallurgical and Chemical Engineering*, vol. 17, 1917, p. 566.

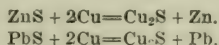
A patent ¹³ was granted in 1915 to Johnson on a proposal to combine his process with the usual retort process. According to this proposal, the zinc ore was treated first in fuel-fired retorts for recovery of the easily reduced portion of the zinc, then the residue was treated in an electric furnace for the recovery of the remainder of the zinc and the lead, copper, gold, and silver. In the usual retort furnace, a large share of the zinc is recovered rapidly at a low temperature, without excessive damage to the retort, and without the necessity for a large excess of reduction carbon. The retort furnace is therefore an efficient agent for this stage of the process. The continuous electric furnace, on the other hand, was particularly adapted for the reduction of low-grade ores containing other valuable metals than zinc, such as the product of the preceding step would be. Theoretically this proposal is sensible; whether it would work out in practice would depend upon the cost of the double handling of the material which would be necessitated.

IMBERT-THOMSON-FITZGERALD PROCESS.

THE ORIGINAL IMBERT PROCESS.

The work of Imbert, Thompson, and Fitzgerald originated in methods proposed by A. H. Imbert for carrying out the precipitation process of treating zinc blende.

Imbert's first proposal ¹⁴ was to treat zinc-lead sulphide ores with metallic copper in externally heated graphite crucibles or retorts similar to those employed for treating zinc-lead-silver drosses from the desilverization of lead bullion. Zinc blende, which by itself is fusible only at very high temperatures, was claimed to melt readily in the presence of copper, and the zinc and lead sulphides were decomposed by the reactions:



The lead sank to the bottom of the retort and the zinc was distilled off and condensed by appropriate means. The copper was recovered by converting the copper matte according to the usual method and used again.

Later the use of copper was abandoned and metallic iron or a ferro-alloy substituted, but since in the ordinary mixture of ore and iron the reaction proceeds very slowly and only at high temperatures, Imbert proposed to dissolve the zinc sulphide in a molten bath and treat the resulting solution with iron. He mentioned various sub-

¹³ Johnson, W. McA., and Hale, E. W., Duplex smelting process, U. S. Patent 1165371, Dec. 12, 1915.

¹⁴ Imbert, A. H., Process of extracting metals from their sulphides, U. S. Patent 807271, Dec. 12, 1905.

stances¹⁵ which would form suitable fluid baths for dissolving the zinc sulphide, among which were mixtures of an alkaline earth oxide and a metallic oxide, such as CaO and FeO, a mixture of 3 parts of iron sulphide with 1 part of iron oxide (this would dissolve 6 parts of zinc sulphide) and later iron oxide (Fe_2O_3) alone, in the proportion of 1 equivalent of iron oxide to 2 equivalents of zinc sulphide. Lead blast-furnace slag also was suggested as a dissolvent. Oxidized zinc ores, or mixtures of oxidized ore with sulphide ores, could be dissolved in a bath of iron oxide and sulphide and treated with iron in the same manner as zinc sulphide.¹⁶

THOMSON-FITZGERALD RADIANT RESISTOR FURNACE.

Imbert soon found that a fuel-fired retort was not suitable for carrying out his process. At this stage John Thomson and F. A. J. Fitzgerald started to design an electric furnace which could be used for the purpose, and from their experiments there developed a great variety of carbon-resistor furnaces, only a few representative ones of which will be described here.

The first group of these furnaces were of the radiant-resistor type, in which the carbon resistor was placed in the furnace chamber just below the arch, and radiated heat downward to the charge.

The first furnace built¹⁷ had an oblong reaction chamber which was roofed over by the resistor. This was built up of carbon rods laid transversely across the top of the reaction chamber, the ends of the bottom row of rods resting on ledges of magnesite brick. The resistor was in contact with terminals at either end of the furnace. The smelting chamber of this furnace would hold 550 pounds of iron.

A three-day test with his furnace gave a radiation loss of 18 kw. with the temperature of the bath at $1,300^\circ \text{C}$. The full working capacity of the furnace was 40 kw., so that the efficiency was about 55 per cent. At the end of this test, the temperature of the furnace was gradually raised to determine at what point the furnace would finally give way. It was found that failure occurred first by the fusion of the magnesite bricks which supported the resistor rods. The arch did not suffer as might have been expected, as the pressure on the lower carbon rods caused less resistance and consequently larger current and higher temperature in the lower part of the re-

¹⁵ Imbert, A. H., Process of treating zinc and lead sulphide ores, U. S. Patent 875578, Dec. 31, 1907; Metallurgical treatment of sulphurous ores by the precipitation process, U. S. Patent 875579, June 6, 1907; Process of reducing zinc ores, U. S. Patent 927857, July 13, 1909.

¹⁶ Imbert, A. H., Metallurgical process of treating oxidized zinc ores by the precipitation process, U. S. Patent 875580, Dec. 31, 1907.

¹⁷ Fitzgerald, Francis A., A new electric-resistance furnace: Trans. Am. Electrochem. Soc., vol. 19, 1911, pp. 273-284.

sistor and a more moderate temperature in the upper part near the arch.

Later another furnace of similar type of 150 kw. capacity was built. In order to obtain the necessary resistance without making the furnace too long, the furnace was built with both terminals at one end, the resistor being in two sections connected with each other at the opposite end of the furnace. One side of each section of the resistor was supported on a ledge running along one side of the furnace, while the other sides were supported on longitudinal arches spanning the furnace chamber.

The inside dimensions of the reaction chamber were 55 inches square by 10 inches deep. The resistor was built of seven layers of rods three-fourths inch in diameter by 20 inches long. Each section of the resistor was about 45 inches long. This furnace gave an efficiency of about 78 per cent at $1,250^{\circ}$ C. and 72 per cent at $1,450^{\circ}$ C.

It was found that the temperature reached in the resistor was too near the working limit of the supporting refractories, so several forms of self-supporting resistor were designed. Some of these were simple arches built up of carbon wedges; others were arches built up of interlocking corrugated carbon plates. A furnace could be arranged to be heated partly by fuel and partly by electricity by building a false arch just under the resistor, thus inclosing it in a separate chamber in which a reducing atmosphere could be maintained to prevent oxidation of the graphite.

The resistance in these resistors is largely in the contacts between the plates or rods and is much greater than that of the graphite alone.

Another interesting adaptation of the above design was for a revolving furnace with a central resistor rod which might be built up of interlocking plates¹⁸ or might take the form of a carborundum tube filled with granular carbon.¹⁹

The Imbert process, using Thomson-Fitzgerald furnaces, was experimented with for several years in Hohenlohehütte, Upper Silesia, and in Australia, but was finally abandoned.²⁰

OTHER THOMSON-FITZGERALD FURNACES.

A later group of furnaces²¹ designed by Thomson for the reduction of zinc ores by carbon were heated by a resistor of granular

¹⁸ Thomson, John, Electric furnace, U. S. Patent 950880, Mar. 1, 1910.

¹⁹ Thomson, John, and Fitzgerald, F. A. J., Furnace, U. S. Patent 950878, Mar. 1, 1910.

²⁰ Ingalls, W. R., Zinc: Mineral Industry, vol. 21, 1912, p. 895.

²¹ Thomson, John, Electric zinc furnace with integral condenser, U. S. Patents 1080862, 1080863, 1080865, 1080866, Dec. 9, 1913; 1086414, 1086415, 1086417, Feb. 10, 1914; 1090427, 1090428, Mar. 17, 1914. Electric zinc furnace with integral compound resistor and compound condensers, U. S. Patent 1086416, Feb. 10, 1914. Universal electric zinc furnace with integral condenser, U. S. Patent 1086418, Feb. 10, 1914. Universal electric furnace combined with means for condensing zinc, U. S. Patent 1090429, Mar. 17, 1914.

carbon inclosed in a grating of refractory rods or coarse pieces of carbon. The resistor was surrounded by and embedded in the ore charge and the zinc vapors passed through the granular carbon of the resistor into a condenser which was built as an integral part of the furnace.

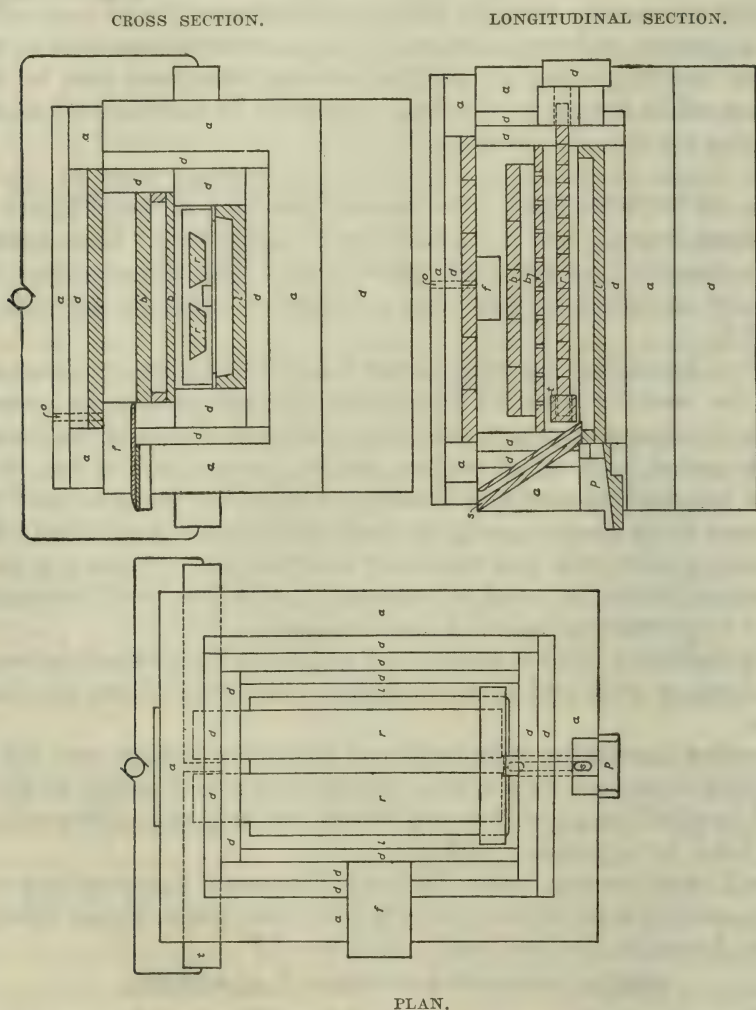


FIGURE 13.—Thomson-Fitzgerald radiant resistor furnace for the redistillation of spelter and scrap zinc—plan, cross section, and longitudinal section (from Fitzgerald, F. A. J., Radiant resistor furnace, Trans. Am. Electrochem. Soc., vol. 36, 1919, pp. 350-352): *a*, Wall of heat-insulating brick; *b*, diaphragms; *d*, inner walls of fire brick; *f*, opening to condenser; *l*, hearth; *o*, hole for pyrometer tube; *p*, tapping spout; *r*, zigzag resistors; *s*, inclined tube for charging molten zinc.

During the war the price of high-grade zinc increased so much that there was incentive for experiments on the redistillation of low-grade spelter or scrap zinc, and Thomson and Fitzgerald adapted their radiant-resistor furnace to this use. The design of a furnace

modified for this purpose is shown in Figure 13 (plan, vertical section, and transverse section). The furnace was about $6\frac{1}{2}$ feet long. The inner walls were of fire brick *d*, outside of which was a wall of heat-insulating brick *a*. The premelted zinc was introduced into the hearth *l* through the inclined tube *s*, the lower end of which extended below the surface of the molten bath. The furnace was heated by zigzag resistors *r*, which were made from graphite slabs with slots cut in them to increase the path of the current. In the later forms of this resistor the slots are filled with alundum cement, thus materially increasing its strength. Above the resistor were two diaphragms *b*. The lower consisted of transverse plates separated by a slight space, and the upper was solid but did not extend to the end of the furnace. The zinc vapor passed between the plates of the lower diaphragm, around the end of the upper one, and thence out of the opening *f* to the condenser. The roof of the furnace was of carbon slabs covered with fire brick and insulating brick. The hole *o* admitted a pyrometer tube. The condenser was a flat box 100 by 50 by 7 cm., with a series of transverse baffle plates. It was heated with a charcoal fire at the start, but as soon as distillation was well started no external heating was necessary, and a good condensation to liquid zinc was obtained.

Experimental work on the redistillation of zinc was carried on for several months with this furnace, with considerable success. The results of a 37-hour run were as follows:²²

Results of 37-hour run with Thomson-Fitzgerald radiant-resistor furnace.

Average power used during distillation.....	kilowatt..	55.2
Total energy for distillation.....	kilowatt hour..	2,051
Minimum temperature of zinc entering condenser.....	°C..	1,040
Average temperature of zinc entering condenser.....	do..	1,225
Average temperature of zinc in premelter.....	do..	550
Weight of zinc condensed (3,494 pounds).....	kilograms..	1,558
Weight of zinc condensed per hour (95 pounds).....	do..	43
Energy per kilogram of zinc.....	kilowatt hour..	1.29
Energy per pound of zinc.....	do..	.59

The total energy consumption per pound of zinc, including pre-melting, during the last 9 hours of the run, was 0.62 kw. h.

Recently Thomson has patented a furnace²³ for the smelting of zinc ore in which the resistor, made of a carbon plate cut into zigzag shape, extends through the center of the furnace, heating partly by conduction and partly by radiation, the ore charge lying in piles on either side of it.

²² Fitzgerald, F. A. J., Radiant resistor furnace: Trans. Am. Electrochem. Soc., vol. 36, 1919, pp. 349-355.

²³ Thomson, John, Electric furnace for reducing oxidized zinc concentrates by carbon, U. S. Patent 1321683, Nov. 11, 1919.

THIERRY FURNACES.

A series of patents ²⁴ have been issued to C. V. Thierry alone, and to C. V. and M. Thierry, on a group of furnaces similar to the granular carbon-resistor furnaces of John Thomson. Thierry's object was the reduction of a concentrated zinc oxide product with carbon, so no provision was necessary for taking care of a solid or liquid residue, except to provide for an occasional cleaning out of the furnace.

One of these furnaces is shown in cross section in Figure 14. In this furnace, *a* is the reaction chamber on the bottom of which lies

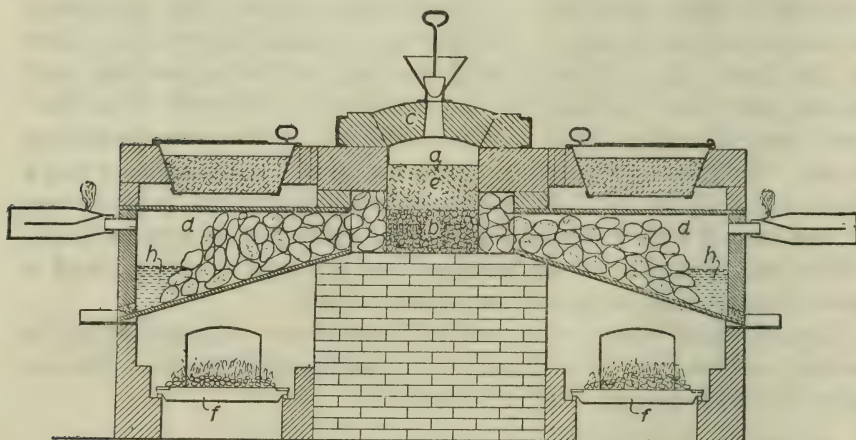


FIGURE 14.—Thierry furnace (U. S. Patent 1030350, June 25, 1912): *a*, Reaction chamber; *b*, carbon resistor; *c*, cover; *d*, condensers; *e*, charge; *f*, grates; *h*, molten zinc.

the granular carbon resistor *b*, making contact with the electrodes at opposite ends of the furnace. The chamber is closed by the cover *c*. The resistor preferably having been previously heated to smelting temperature, the charge *e* of zinc oxide and reducer is introduced through the hoppers onto the surface of the bed of carbon. The zinc vapor produced by the reduction of the zinc oxide passes through the carbon of the resistor and into the condensers *d*, which also contain granular carbon. The condensers are arranged to be preheated by means of the grates *f*. This furnace is claimed to have given good yields of metallic zinc.²⁵

THE TAYLOR SHAFT FURNACE.

Edward R. Taylor has designed a series of shaft furnaces for smelting ores. One of these intended for the smelting of zinc ores

²⁴ Thierry, C. V., Metallurgy of zinc, U. S. Patents 1030349, 1030350, June 25, 1912. Electric zinc furnace with integral condenser, U. S. Patents 1110359, Sept. 15, 1914; 1122663, 1122664, Dec. 29, 1914.

²⁵ Ingalls, W. R., Zinc: Mineral Industry, vol. 21, 1912, pp. 894, 895.

was described by him in a paper read before the American Electrochemical Society in 1907.²⁶

The following description is quoted from this paper:

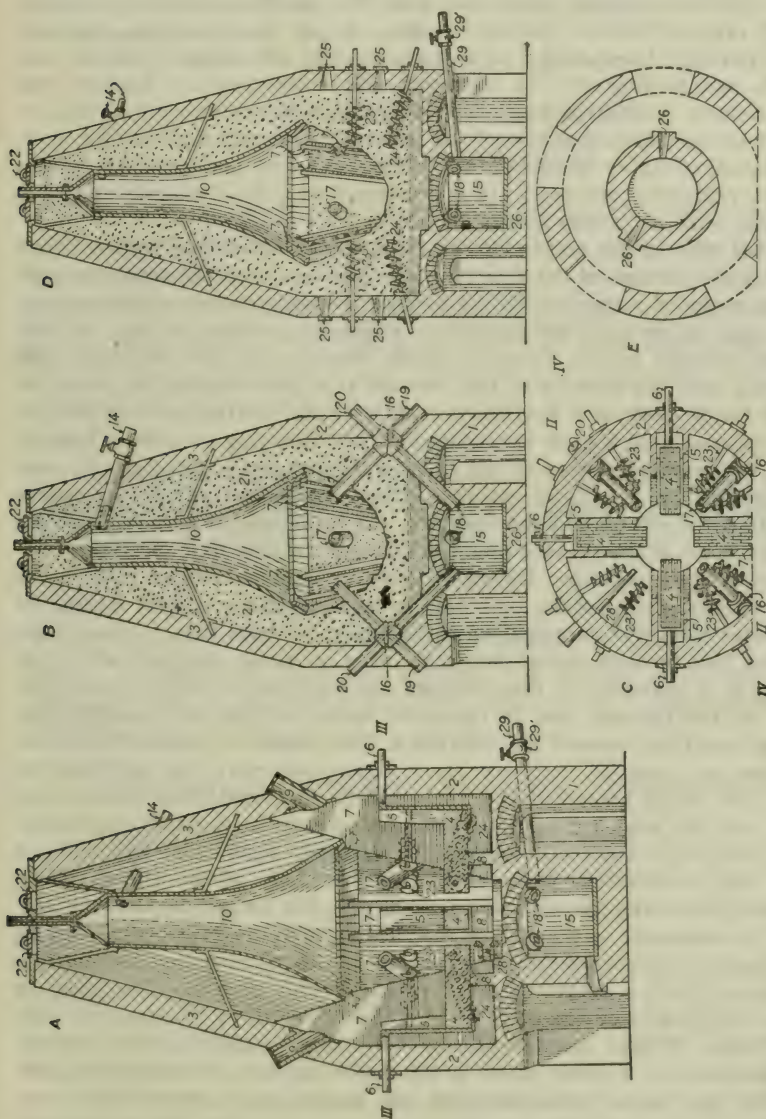


FIGURE 15.—The Taylor shaft furnace (Taylor, Edward R., A closed electric furnace for reducing and distilling metals from their ores, Trans. Am. Electrochem. Soc., vol. 11, 1907, p. 296) : 1, Base ; 2, reducing chamber ; 3, shaft ; 4, horizontal electrodes ; 5, 6, metallic conductors ; 7, partitions inclosing electrode ; 8, piers ; 9, charging aperture ; 10, hood ; 14, pipe ; 15, chamber ; 16, conduits ; 17, 18, 19, 20, legs ; 21, insulating material ; 22, apertures ; 23, 24, screws ; 25, 26, 27, 28, tap holes ; 29, openings.

The furnace structure (fig. 15) comprises a base 1, reducing chamber 2, and shaft 3, the latter for the introduction of the charge and the removal of certain of the gases formed in the operation.

²⁶ Taylor, Edward R., A closed electric furnace for reducing and distilling metals from their ores : Trans. Am. Electrochem. Soc., vol. 11, 1907, pp. 295-301.

The base may be formed of a series of masonry arches supporting the superstructure and inclosing a collecting chamber 15. Within the furnace and above the base are horizontal electrodes 4 of carbon supported by masonry piers 8 and provided with metallic conductors 5 and 6. On either side of each electrode are vertical retaining walls, and near the upper portion is a charging aperture 9 provided with a suitable closure. In the upper portion, and supported in part by these walls, is a hood, or bell 10, flaring outward and downward, and serving for the collection and withdrawal of some of the gaseous products of the reaction. This hood may be constructed of iron or steel for moderate temperature work, or suitable refractory material for operations demanding higher temperatures, or exposed metal portions may be protected with refractory material. 14 represents a pipe for the removal of the lighter gases, such as carbon monoxide (CO).

Within the base of the furnace is the chamber 15, referred to before, which serves for the collection of the volatile metal portions resulting from the reaction. This chamber connects with the furnace proper by means of the conduits 16, which may be made of fire-clay tubes or graphitic carbon, or may be built within the masonry of the furnace. These conduits are provided with legs 17 extending into the interior of the furnace at a point somewhat above the electrodes (or so they will not be covered with the charge passing through the furnace) and with legs 18 extending into the collecting chamber. Legs 19 and 20, in prolongation of legs 17 and 18, pass through the walls of the furnace, and are provided with suitable closures. These extensions afford opportunity for inspection and cleaning of the conduits. In some cases it may be desirable to withdraw some of the furnace products through the legs 19.

Adjacent to these conduits, and preferably arranged on two or more levels of the furnace, are a plurality of screws or equivalent mechanical devices 23 and 24, constructed and arranged to continuously, or from time to time, force portions of the furnace charge into the reaction zone. As best shown in Figure 15, C, a plurality of these screws are arranged radially around the periphery of the furnace, and in operation serve to force the material into the field of reaction between the electrodes. As shown in Figure 15, D, the same screws are mounted in two or more superposed rows, the screws 23 in the upper row being bladed at their inner ends only, while the screws 24 of the lower row are bladed from their inner ends nearly or quite to the furnace walls, and inclined upwardly toward the reaction zone. The purpose and effect of this construction is to insure that portions of the charge which descend along the periphery of the furnace, past the screws 23, shall be conveyed by the screws 24 to the field of reaction through the surrounding charge.

Ores adapted to being worked in a furnace of this construction are fed into the furnace at the top apertures 22 provided for that purpose. They may be previously mixed with carbon or other reducing material, or, if necessary, with suitable fluxes. Broken carbons may be fed upon the electrodes as required through openings 9. Several years in the use of broken carbons in the reinforcement of the main carbon electrode has amply demonstrated their great usefulness, both for regulating the current and prolonging the life of the main carbon electrode. For an expenditure of \$175 for the main carbons we have been enabled to produce 2,000,000 pounds of bisulphide of carbon in the electric furnace. In much electric furnace work the cost of carbons is a large item in the cost of the finished product, which is here shown to be almost infinitesimal, certainly not one of the larger of the fixed charges of running a plant.

As the ore is moved forward into the reaction zone it is replaced by fresh portions from the periphery of the furnace. Any slag or nonvolatile reduction products may be drawn off from time to time through the tap holes 27 and 28.

Provision is made for the lighter gases to pass through the hopper 10 and the pipe 14, while the heavier volatile metals may pass more readily through 16 to the receiving chamber 15, entering by the pipes 18, and such gases as are not condensed under such circumstances out through the pipe 29. By controlling the openings 14 and 29, the direction and movement of these gases and vapors may be directed and controlled.

The metal collecting in 15 may be tapped off through the tap holes 26.

COTE AND PIERRON PROCESS.

EARLY EXPERIMENTS.

E. F. Cote and P. R. Pierron at first directed their attention largely to the direct treatment of raw zinc sulphide ores with metallic iron in an electric furnace. In this way they hoped to do away with some of the difficulties of condensing zinc vapor in the presence of the large amounts of carbon monoxide and considerable proportions of CO_2 formed when the ore is first roasted and then reduced with carbon. They found, however, that due to the moisture and carbonates in the ore and the air introduced into the furnace with the charge, considerable quantities of harmful gases were still produced. They therefore introduced a column of incandescent carbon between the furnace and the condenser to reduce the carbon dioxide and water vapor to carbon monoxide and hydrogen.

Their various patents are mainly on different designs for combining the electric furnace and the column of carbon in practical form. In some of these the carbon was heated electrically and in others it was so managed that the heat of the gases going into it would keep it at the necessary temperature.

FLEURVILLE'S DESCRIPTION OF THE PROCESS IN 1908.

In 1908, Eugene Fleurville published a comprehensive account of the development of the Cote and Pierron process and a description of the furnace then in use.²⁷ Figure 16 is a diagram of the furnace. The bottom and side walls of the crucible *a* were made of graphite, incased in sheet metal, and were connected to one pole of the electric circuit. There was a projection *b* in the center of the furnace bottom, directly under the electrode *c*, so that the current flowed between this projection and the electrode. The arch *d* was of refractory brick. The holes *e* served for the introduction of the charge and could be closed by refractory stoppers. The zinc vapor passed through the outlet *g* and passage *h* to the condenser *j*, which was a vertical chamber filled with pieces of carbon *i*. The condenser was tapped through the hole *o*. Air could be admitted at *k* and drawn into the stack *m*, allowing the carbon in the upper part of the cham-

²⁷ Fleurville, Eugene, Electric zinc smelting: *Electrochem. and Met. Ind.*, vol. 7, November, 1909, pp. 468-472.

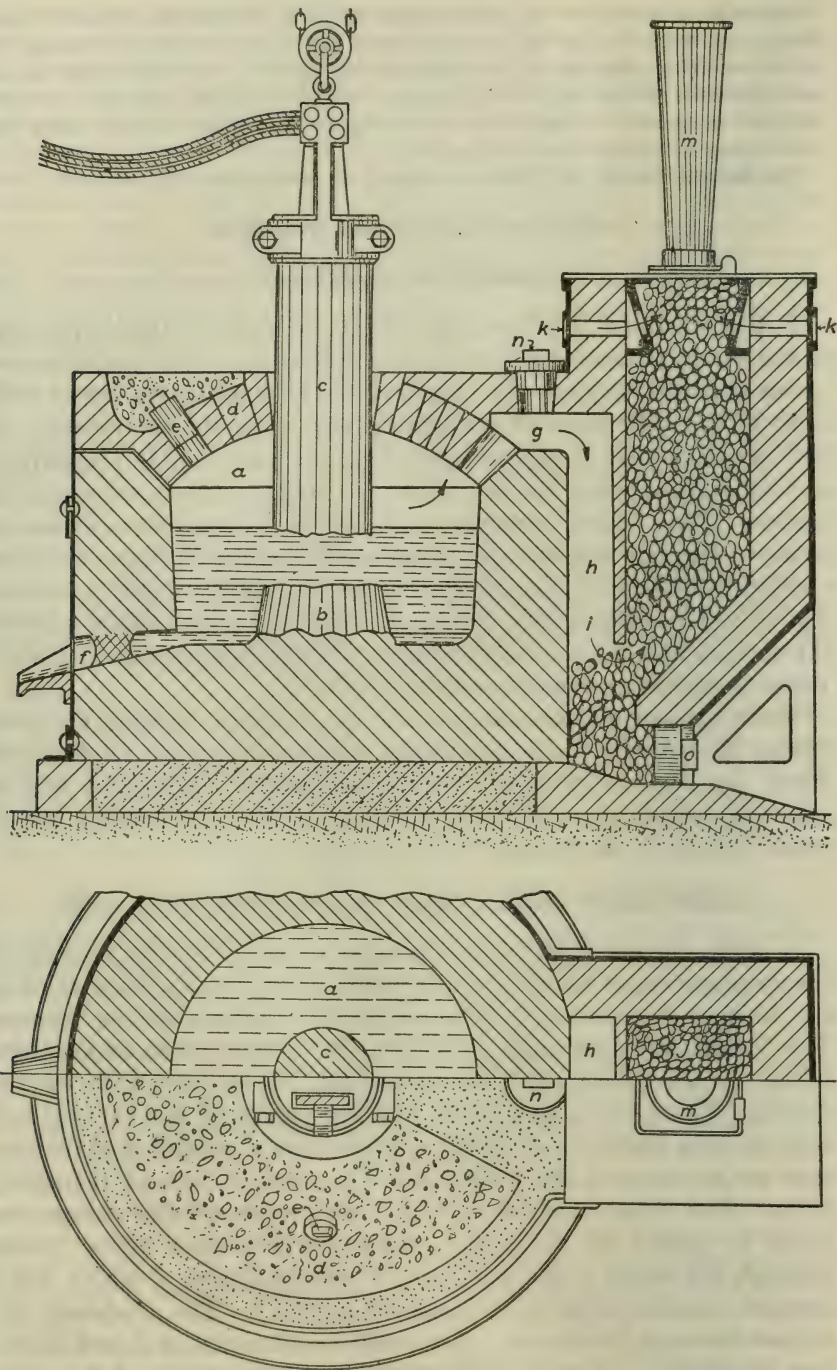


FIGURE 16.—Cote and Pierron furnace (French Patent 385018, 3d addition, Nov. 9, 1908) :
a, Crucible; *b*, projection from bottom electrode; *c*, electrode; *d*, arch; *e*, charge holes;
f, spout; *g*, outlet; *h*, passage for zinc vapors; *i*, carbon; *j*, condenser; *k*, opening for
admission of air; *m*, stack; *n*, opening for cleaning channels; *o*, hole for drawing zinc.

ber to be heated to a red heat. The downward passage of the carbon was regulated so that the heat of the gases coming into the condenser, with the heat liberated by the condensation of the zinc, would keep the condenser at the proper temperature for the reduction of "zinc mist" and condensation of the zinc. The opening *n* was provided for cleaning the channels *g* and *h* in case of clogging. The condenser could have been heated electrically if necessary.

The charge was first dried on the roof of the furnace. The previous charge having been tapped out, the upper electrode was lowered into contact with the projection *b*. The charge was then introduced through the holes *e*, the holes again closed, and a new charge placed on top of the furnace. The electrodes being in contact with each other, practically no current passed through the charge, which was heated only by contact with the electrodes. The lead was precipitated at this stage and tapped out. The door *o* at the bottom of the condenser was left open during this time and the pieces of carbon allowed to slide down enough to clean the condenser. At the same time, the carbon was heated to the proper temperature by the gases produced in the furnace. After the lead had all been precipitated and removed from the furnace the tap hole and condenser door were closed and the electrode *e* gradually raised so that the charge was heated by resistance and arc heating. The zinc liberated and the iron sulphide formed settled into the annular ring around the lower electrode projection. Finally, by simple arc heating, the last of the zinc was set free and the slag was melted and tapped out, leaving the furnace ready for the next charge.

The iron sulphide was to be sold to sulphuric acid manufacturers, partly compensating for the cost of the metallic iron used.

The hot zinc vapor going into the condenser maintained the lower part of its column of carbon at a sufficient temperature to reduce any zinc oxide in the vapor. As the vapor passed up into the cooler regions, it was condensed to liquid zinc and trickled to the bottom where it was tapped off. Any dust or foreign material tending to clog the condenser was removed by raking out some of the carbon from time to time and fresh carbon was introduced at the top to replace it. The small amount of carbon monoxide passed out through the stack *m*. Incomplete condensation of gas would be shown by escaping fumes of zinc oxide; then the temperature of the condenser could be lowered by increasing the speed of the passage of the carbon down the condenser or by cutting down the power to the furnace crucible.

The first experiments with this process on a semicommercial scale were made in 1906 at Lyons, France. The furnace consisted of a cylindrical crucible of 40 cm. inside diameter and 45 cm. high, built

up of magnesia brick held together by a metal casing. The electrodes were placed vertically along the central axis of the crucible and charge hopper. The furnace worked somewhat on the same principles as the one above described. The experiments with this furnace were intended chiefly to work out the chemical problems of the precipitation reactions, power consumption, metal losses, and electrode consumption. No particular effort was made to obtain liquid zinc.

The Société des Fonderies Electriques was formed in 1907 to continue the experiments on a larger scale. The plan was first to study the suitability of various types of furnaces for the process and then to determine the cost of operation and commercial possibilities of a 500-horsepower plant. It was intended to treat low-grade zinc ores for the recovery of zinc white, leaving the problem of condensation of spelter till later. After successful completion of this step, a 1,200 to 1,500 horsepower plant was to be built for the making of zinc white. As experience was acquired in the handling of large volumes of zinc vapor, the production of liquid spelter was to be increased until practically all of the zinc was recovered as metal.

At the time Fleurville's article was written, the first step of this program had been successfully completed. Furnaces of many types were tried, the one finally adopted being of the type described above. The author notes that the manufacture of zinc oxide involves its own problems which may or may not be less difficult than the problems of zinc condensation.

Following are the results of a typical furnace run :

Total time of series of experiments, 240 hours.

Weight of ore smelted during that time, 14,560 kilograms.

Average zinc content, 43.6 per cent.

Weight of iron mixed with ore, 6,235 kilograms.

Weight of fluxes added to the charge, 3,480 kilograms.

Mean current, 4,300 amperes; voltage, 40; power factor, 0.80.

Weight of zinc oxide obtained, 6,730 kilograms.

Metallic zinc condensed before passage to the combustion apparatus, 182 kilograms.

Zinc content in the slag and iron sulphide, 2.7 per cent.

Consumption of electrodes, 193 kilograms.

Electric energy cost \$13 per horsepower year.

Electrodes cost \$10.40 per 100 kilograms.

Iron cost, \$12 per ton; lime, \$3 per 1,000 kilograms.

The workmen were paid an average of 60 cents per day.

Under these conditions the cost of 4,912 kilograms of oxide and metallic zinc was \$479, not including general expenses but including maintenance of the furnaces. Adding the cost of packing and loading into cars brought the total cost to \$526. The oxide was sold for \$640.

For the production of commercial zinc white, the use of iron was later given up because some of the iron volatilized, discoloring the zinc oxide. Instead of iron, lime and carbon were used. These react with the blende according to the reaction $\text{ZnS} + \text{CaO} + \text{C} = \text{CO} + \text{CaS} + \text{Zn}$. The results for a furnace run using this method of reduction are given and are similar to those obtained when iron was used. The zinc oxide produced analyzed 98.6 to 99 per cent ZnO , 0.4 per cent moisture, 0.2 to 0.3 per cent iron, and 0.5 to 0.7 per cent silica, lime, and other impurities. The ore was free of other metals which would contaminate the oxide.

TEST REPORTED IN 1914.

In 1914 results were reported of a test in which the zinc was obtained as spelter.²⁸ In 72 hours the furnace smelted 9,560 kg. of blende, assaying 34.8 per cent zinc, mixed with 2,970 kg. of iron. The electric energy used was 16,285 kw. h. There were produced 2,800 kg. of spelter, assaying 99.06 per cent zinc. The iron sulphide and slag weighed 9,730 kg. and ran 1.85 per cent zinc. Electrode consumption was 110 kg. The daily capacity of the furnace was placed at 3,200 kg. of ore at 1,700 kw. h. per metric ton (1,574 kw. h. per ton of 2,000 pounds). Two men were required to operate the furnace. The cost per metric ton of 34 per cent to 38 per cent ore was \$9.52, or \$8.63 per ton of 2,000 pounds.

COMPOUND SMELTING FURNACE AND CONDENSER.

In 1917 G. Flusin reported²⁹ that a plant was under construction at Maurienne, France, which was to have four 500-horsepower furnaces with a daily capacity of 4 metric tons of ore per furnace.

The process as used at that time consisted of the treatment of raw blende or mixed sulphides with lime and carbon in a "compound furnace." This was really a combination of two furnaces. The charge was smelted in the first furnace, of the arc-resistance type. The second furnace, of the indirect-resistance type, received the zinc in the form of droplets and zinc dust directly from the first furnace, and with the addition of a small amount of heat energy, revolatilized it as pure zinc vapor which was condensed to liquid spelter. The power consumption was estimated at 2,000 to 2,200 kw. h. per metric ton of 35 per cent zinc ore. Seventy to 75 per cent of the power was used in smelting and 25 to 30 per cent in redistillation. The slag assayed 1.5 per cent zinc and the spelter 99.93 per cent. The total

²⁸ Engineering and Mining Journal, The Cote-Pierron electric zinc-smelting process: Vol. 97, Apr. 11, 1914, p. 756.

²⁹ Flusin, G. [Recent development of the Cote and Pierron electric zinc-smelting process]: Bull. Tech. Suisse Romande, vol. 23, No. 17, 1917, p. 233; abstract, Met. and Chem. Eng., vol. 18, Jan. 1, 1918, p. 17.

loss of zinc was less than 10 per cent. Electrode consumption was 12 kg. per metric ton.

The compound furnace mentioned by Flusin is evidently of the type described in Cote and Pierron's United States Patent 1,184,520 of May 23, 1916. Figures 17 and 18 illustrate this furnace. The smelting furnace, Figure 17, is similar to the earlier one described by Fleurville. The vapors from this pass into the chamber *h* of the

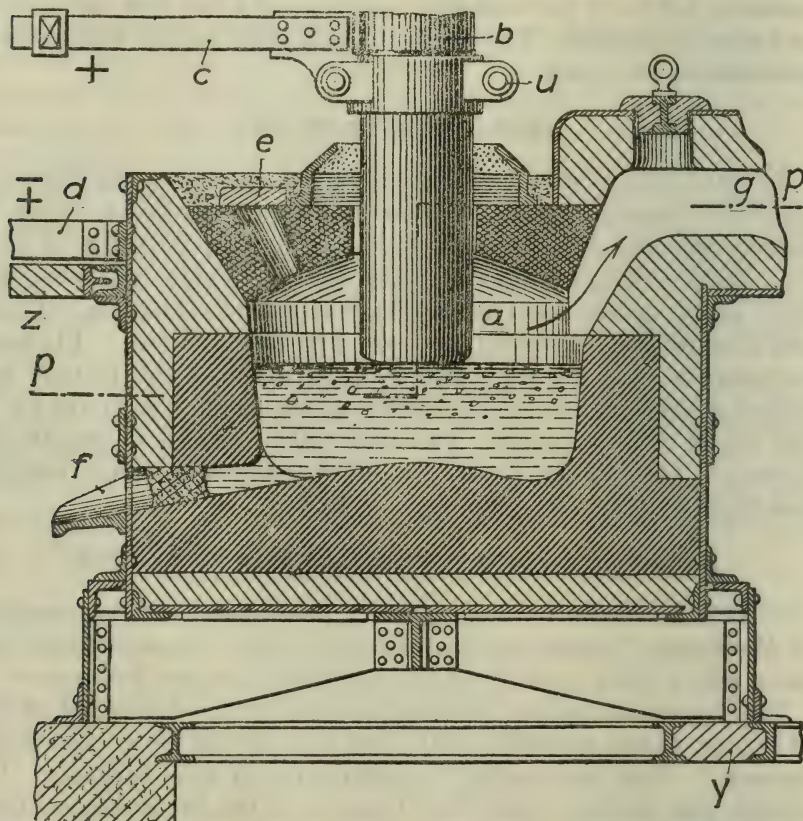


FIGURE 17.—Cote and Pierron smelting furnace (U. S. Patent 1184520, May 23, 1916) : *a*, Chamber; *b*, electrode; *c*, *d*, busbars; *e*, charge hole; *f*, spout; *g*, passage for zinc vapor.

refining furnace, Figure 18. Here most of the zinc condenses as dust and droplets and falls upon the bed of carbon below. Any zinc vapor remaining is condensed in the carbon column *j*. The column of carbon below the condensing chamber is caused to move downward, carrying the zinc powder with it, by removing carbon through doors at the bottom and replacing it with the carbon raked out of the column *j*, which in turn is replaced by a fresh supply. As the zinc powder is carried below the narrowed region *k* of the column, it is heated by an electric current between the graphite ring *l* and the electrode *s* and revolatilized. The pure zinc vapor thus produced

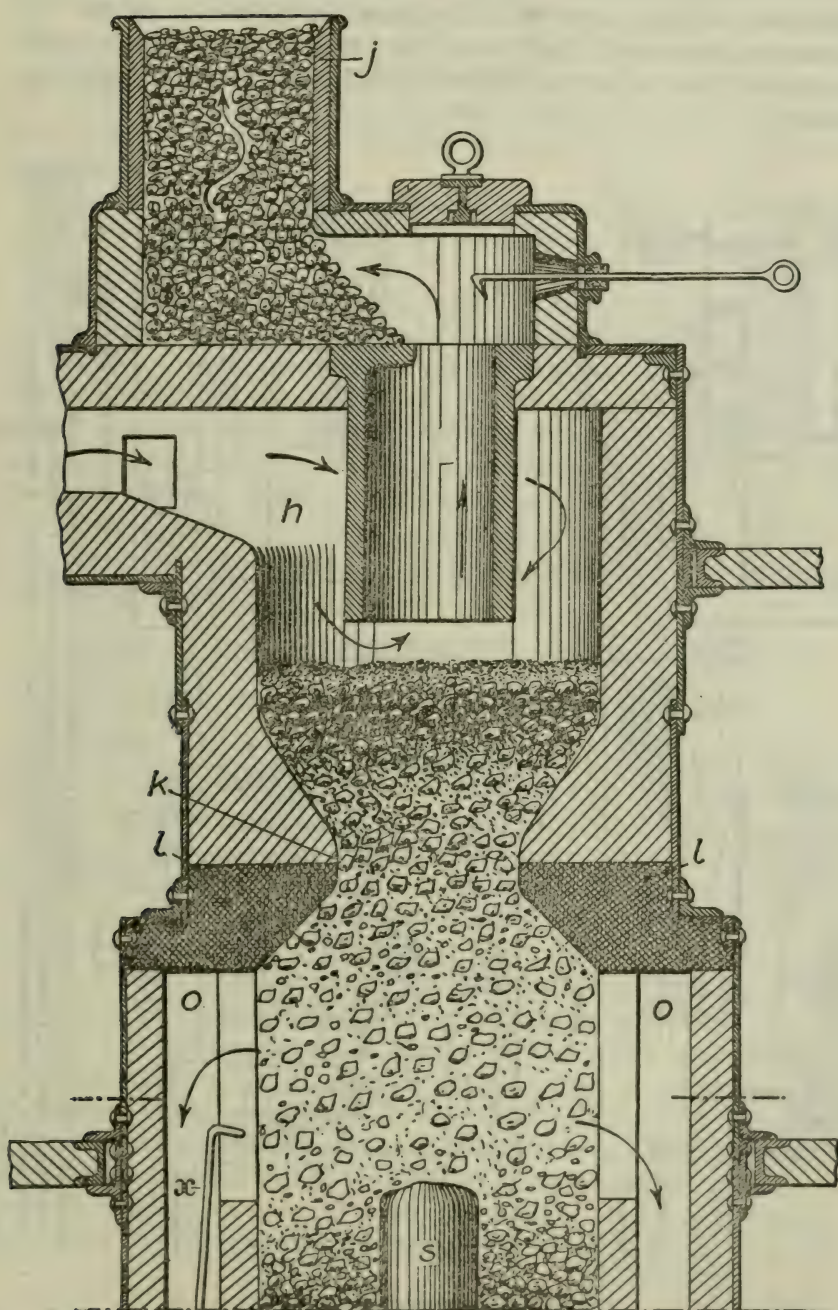


FIGURE 18.—Cote and Pierron refining furnace (U. S. Patent 1184520, May 23, 1916) :
h, Condensing chamber; *j*, carbon column; *k*, narrow region of column; *l*, graphite
 ring; *o*, annular ring serving as condenser for refined zinc vapor; *s*, electrode.

passes through the slits and into the annular ring *o*. Here it is condensed to liquid metal and is collected at the bottom of the condenser compartments. Any liquid zinc which passes down the carbon column without being revolatilized is collected at the bottom of the column.

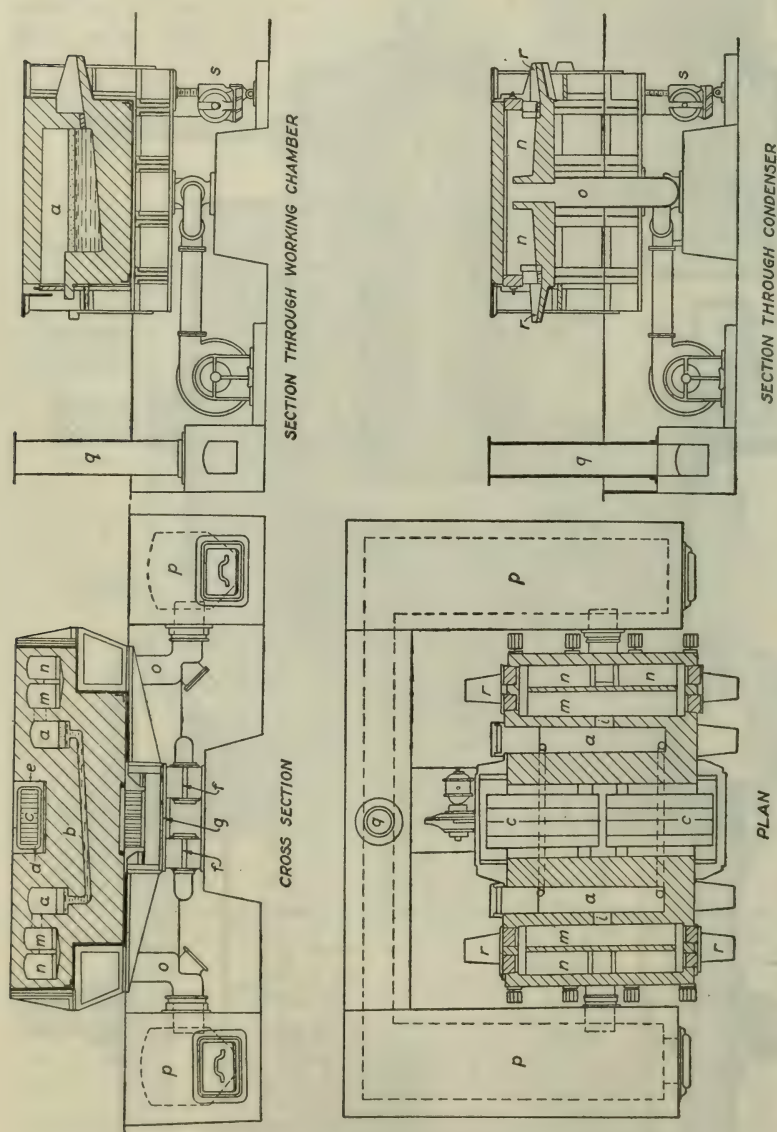


FIGURE 19.—Gin induction furnace (British Patent 28638, 1906) : *a*, Working chambers ; *b* conduits ; *c*, double magnetic circuit ; *d*, insulated copper coils ; *e*, water-cooling channels ; *f*, trunnions ; *t*, passages to condensing chambers ; *m*, *n*, condensing chambers ; *o*, conduit for gases from condenser ; *p*, dust chamber ; *q*, stack ; *r*, spout ; *s*, jackscrew.

GIN INDUCTION FURNACE.

One of the early furnaces suggested for the electrothermic production of zinc was the induction furnace of Gustave Gin shown in Figure 19. The two parallel channels *a* formed the working cham-

bers of the furnace, and with the conduits *b* of circular cross-section made up the secondary circuit of a transformer. The two insulated copper coils *d*, connected in series, wound on the upper horizontal branches of the double magnetic circuit *c* formed the primary circuit of the transformer. The magnetic core and primary winding were protected from the hot furnace walls by air cooling and by the water-cooled channels *e*.

The whole furnace was contained in an iron shell resting upon two iron trunnions *f*. The lining was of burnt dolomite tamped in with a pitch binder.

A bath of iron or other metal in the working channels and connecting conduits served to carry the secondary current and produced the heat for smelting the charge which was spread upon its surface. The iron could take part in the reduction of the zinc, if desired, or if roasted ore and carbon were charged the iron would simply serve as resistor. The zinc vapor and gases produced passed through the channel *i* into the condensing chambers *m* and *n*, and thence out through the conduit *o* to the dust chamber *p* and stack *q*. The furnace could be tipped by the motor-driven jackscrew *s* to facilitate the tapping operation, pouring at *r*.

WORK OF F. T. SNYDER AND THE CANADA ZINC CO.

DESCRIPTION OF SOME OF THE SNYDER FURNACES.

From 1906 to 1909, patents were granted to Frederick T. Snyder on several furnaces and processes for smelting zinc ores electrically.³⁰

These included an induction furnace, two vertical shaft furnaces, and several forms of slag-resistance furnaces. Snyder appreciated the beneficial effect of prereduction of the charge and of a concentrated zinc vapor in obtaining condensation to liquid zinc, and took these into consideration in his design.

Two of these furnaces, one of vertical-shaft type and one of the slag-resistance type, are illustrated in Figures 20 and 21.

The shaft furnace shown in Figure 20 had a fire-brick base, *a*, inclosed in an iron shell, *a'*, held in place by lugs, *p*, surmounted by the water-jacketed rectangular furnace shaft *b*, which was left open at the top to receive the charge and permit the escape of uncondensed vapors. The crucible was to be kept full of molten metal, matte, and slag. A longitudinal row of three electrodes, *c*, projected vertically downward into the furnace far enough for the ends to dip into the

³⁰ Snyder, F. T., Metallurgical process, U. S. Patents 814810, Mar. 13, 1906; 859135, July 2, 1907. Electric furnace, U. S. Patents 825359, July 10, 1906; 859136, 859137, July 2, 1907; 942110, Dec. 7, 1909. Process of treating ores, U. S. Patent 834644, Oct. 29, 1906. Smelting process, U. S. Patents 859132, 859134, July 2, 1907. Smelting furnace, U. S. Patent 859133, July 2, 1907. Process of treating zinc ores, U. S. Patent 933133, Sept. 7, 1909.

molten conducting material in the crucible. These electrodes were star connected to a three-phase, alternating current, the neutral point being connected to the metal in the crucible of the furnace.

The hopper *d* was arranged to feed powdered coke immediately around the electrodes, the main body of the charge being fed between the outer edges of this hopper and the side walls of the furnace. The consumption of electrodes was thus considerably reduced.

In operating the furnace the top of the charge was maintained at a temperature low enough to condense zinc vapor and prevent its

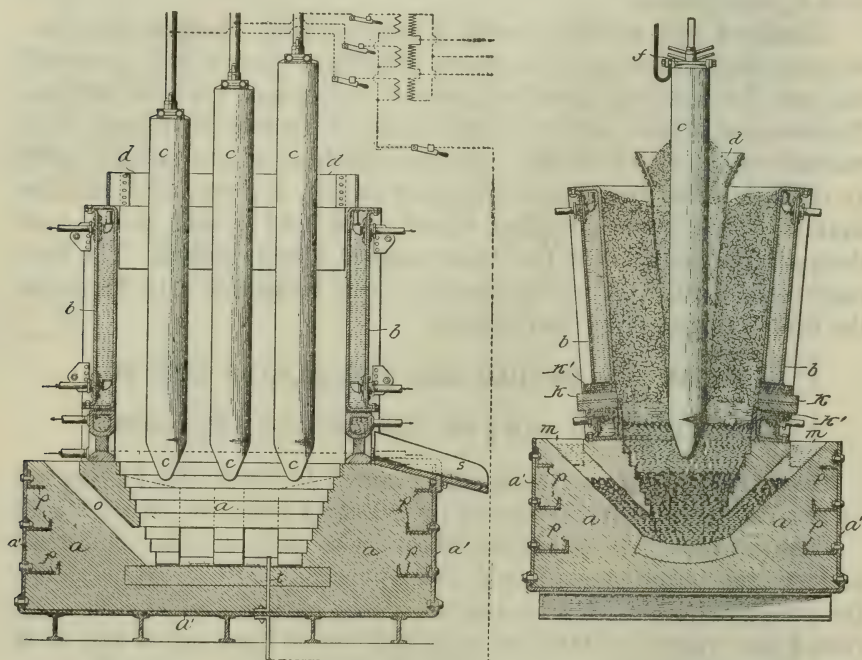


FIGURE 20.—Snyder shaft furnace (U. S. Patent 942110, Dec. 7, 1909): *a*, Fire-brick base; *a'*, iron shell; *b*, furnace shaft; *c*, electrodes; *d*, hopper; *k*, drains for liquid zinc; *k'*, water jackets; *m*, zinc wells; *o*, lead well; *p*, lugs; *s*, slag tap.

escape. As the charge traveled down the shaft it was gradually heated and the easily reducible oxides were reduced to metal. The gases from this preliminary reduction, which would otherwise dilute the zinc vapor formed later, were thus eliminated before the charge reached the temperature for the reduction of zinc oxide. When the charge reached the smelting zone the zinc oxide was reduced, the resulting CO gas escaped from the top of the furnace while the zinc vapor condensed in the cooler portions of the charge and was returned with it to the smelting zone. In this way the charge was progressively enriched until a large part of the zinc condensed to liquid metal against the water jackets. It then flowed down the sides of the furnace and escaped through the drains *k* provided for the

purpose. The shelf, or crust, of slag formed around the sides of the furnace just below the drains by the cooling effect of the water jackets *h'* assisted in diverting the liquid zinc into the drains.

The remainder of the charge, after the zinc was distilled off, melted to slag, matte, and bullion and settled into the crucible. The slag and matte were drawn off through the slag tap *s* at the top of the crucible while the lead was ladled out of the lead well *o*, which communicated with the bottom of the crucible.

The liquid zinc that drained from the tubes *k* was collected in well *m* in the base of the furnace, which could be connected to the lower part of the crucible. These wells served to permit the refin-

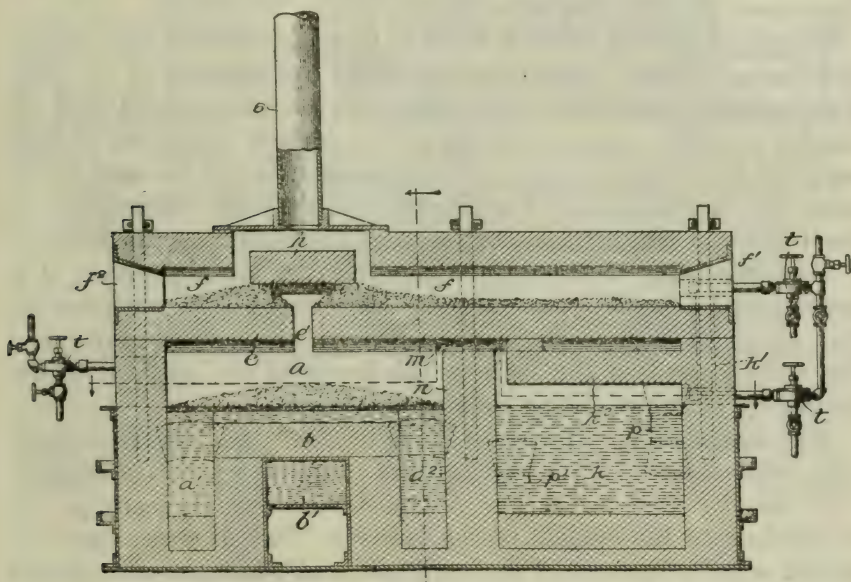


FIGURE 21.—Snyder slag-resistance furnace (U. S. Patent 859137, July 2, 1907) : *a*, Furnace chamber; *a'*, *a²*, chambers filled with molten lead; *b*, bridge wall; *b'*, water jacket; *e*, roof of smelting chamber; *e'*, passage between preheating chamber and furnace; *f*, preheating chamber; *h*, cover; *k*, condenser; *k'*, flues; *k²*, roof; *m*, opening to condenser; *n*, partition; *p*, *p¹*, passages for lead and refined spelter; *s*, stack; *t*, oil burners.

ing of the zinc, as the lead contained could settle out and join the main body of the same metal which was contained in the crucible.

In the resistance furnace illustrated in Figure 21, *a* represents the furnace chamber proper and *e* its roof. This was divided by the bridge wall *b*, cooled by a circulation of water in the water jacket *b'*. The two chambers *a'* and *a²* formed by this bridge wall were filled with molten lead and each chamber communicated with the outside of the furnace by two lead wells, on opposite sides of the furnace. The electrodes were dipped into the lead wells on one side of the furnace while the lead wells on the other side permitted the metal

to be ladled out as fast as it was produced, keeping the metal inside the chambers at the desired level. The top of the bridge wall was covered with a layer of slag connecting the two chambers containing lead and providing a passage between them for the electric current. The charge was so proportioned that this slag should have a high electrical resistance, a low dissolving power for zinc oxide, and a formation temperature between the temperatures of volatilization of zinc and lead (between $1,000^{\circ}$ and $1,100^{\circ}$ C.). A high-lime slag of about the composition 30 per cent CaO, 30 per cent FeO, and 40 per cent SiO_2 was mentioned as being suitable.

The heat required for smelting was produced by the passage of the current through this layer of slag.

On top of the slag rested a layer of incandescent coke and on this the charge proper. Since the formation temperature of the slag was regulated to slightly over $1,000^{\circ}$ C. it was assumed that as long as a sufficient amount of charge was present any excess of heat supplied after this temperature was reached would be used up in reducing zinc oxide and forming slag, so that the temperature of the furnace would not rise high enough to volatilize much lead.

The preheating chamber *f* for roasting the ore and giving it a preliminary reduction was connected with the smelting chamber by the opening *e'* over which was the cover *h*. This cover overlapped the opening so that the latter could be sealed by heaping up the materials in the preheating chamber around it.

The condenser *k*, large enough to be used as a refining chamber, was connected with the furnace chamber by the passage *m* over the partition *n*. It was provided with two passages, *p* and *p'*, leading to the outside of the furnace. One of these was connected with the bottom of the condenser and permitted the removal of any lead which separated from the spelter, while the other, opening into the condenser near the top, provided for the removal of the refined spelter. The gases remaining after the condensation of the zinc vapor were led through the flues *k'* to the preheating chamber where the CO was burned to furnish heat for the preliminary reduction. The gases from the preheating chamber passed out through the stack *s*. Oil burners *t* were provided for heating up the cold furnace and melting the slag when starting after having been shut down for some time. These burners were shut off and the openings plugged with clay while the regular smelting operation was being carried on.

In operating the furnace, the crushed and, if necessary, partly roasted ore, was fed into the preheating chamber and the roasting completed. The proper proportions of reduction material and fluxes

were then added, and after the reduction had been carried as far as possible without volatilizing the zinc, the charge was introduced into the smelting chamber through the opening e' and the opening sealed. The zinc was there reduced and volatilized and entered the condenser with the CO gas produced. The lead, gold, and silver settled to the bottom of the electrode chamber as bullion and were drawn off

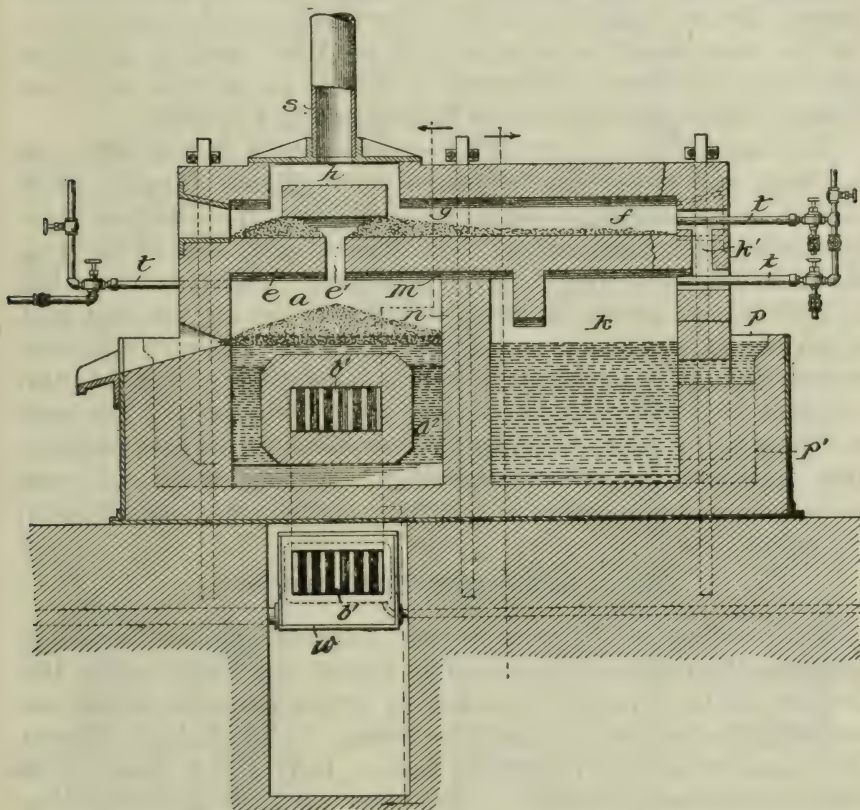


FIGURE 22.—Snyder induction furnace (U. S. Patent 859137, July 2, 1907): a , Furnace chamber; a^2 , chambers filled with molten lead; b' , transformer core; c , roof of smelting chamber; e' , opening between preheating chamber and furnace; f , preheating chamber; h , cover; k , condenser; k' , flues; m , passage to condenser; n , partition; p , zinc well; s , stack; t , oil burners; w , primary winding.

through the lead wells, and the other constituents of the ore formed slag which was tapped out as necessary, always leaving enough to carry the heating current.

The zinc and any lead volatilized were condensed and separated in the condenser, while the CO gas passed on into the upper chamber and was burned to preheat the succeeding charge.

Figure 22 shows the same furnace arranged as an induction furnace. The transformer core is represented by b' , and w is the pri-

mary winding. The secondary circuit is formed by the molten metal and slag contained in the smelting chamber, this material extending completely around the core.

EXPERIMENTS AT VANCOUVER AND NELSON, BRITISH COLUMBIA.

Snyder began experimental work with small furnaces at Vancouver, British Columbia, in 1905. Several types of furnace, 50 to 150 kw. size, were tried during 1905 and 1906, leading up to the shaft style of furnace, somewhat similar in shape to a lead blast furnace, which was later used at Nelson, British Columbia.

The Canada Zinc Co. erected a 10-ton plant at Nelson in 1908, with buildings and equipment designed for an ultimate capacity of 30 tons of ore per day, financial aid being obtained from the provincial government for the purpose. The furnace used was of the type shown in Figure 20 (see p. 42). This plant started operations late in 1908 on ores averaging about 40 per cent zinc, 10 per cent lead, $1\frac{1}{2}$ per cent copper and 12 ounces silver per ton,³¹ and very optimistic reports were given out as to the success being attained, but the plant was shut down about a year later and work discontinued. The work at Nelson was a series of mechanical troubles with furnace construction, operation, and electrode breakage, and none of the runs seem to have been long enough to test the metallurgical possibilities of the process.

LATER EXPERIMENTS AT CHICAGO.

Just previous to the war, Snyder designed another furnace to overcome the faults of those used at Nelson (see Fig. 23). In this furnace a bath of lead, 6, forming the bottom electrode, makes contact with the bus bar 9 through the passage 7 and block 8. The tap hole is shown at 10. Above the furnace body 1 is a frame 12, in which is mounted the roof 13. The upper electrode 14 is held by an arm which can be raised and lowered. The roof is provided with a plurality of openings around its periphery for introduction of the charge, and is arranged so that a considerable supply of ore can be stored upon it. Below each of the charging ports 17 is a reciprocating feeder, 18, to push the charge into the smelting chamber as it gradually works down through the opening.

The zinc, volatilized in the smelting chamber, condensed in the incoming charge and trickled down into the troughs 21, from which it was conducted to a collecting trough, 22, and withdrawn through the outlet 23. When such fine ore was being smelted as to prevent the condensed zinc from percolating through it, the ore and coke

³¹ Electrochemical and Metallurgical Industry, Electric zinc in Canada: Vol. 7, 1909, p. 139.

were charged separately as shown, the zinc finding an outlet through the coarser coke.

A 1,500-kw. furnace of this type was built in Chicago and ran for a short time on zinc ores. Plate I, A, is a photograph of this fur-

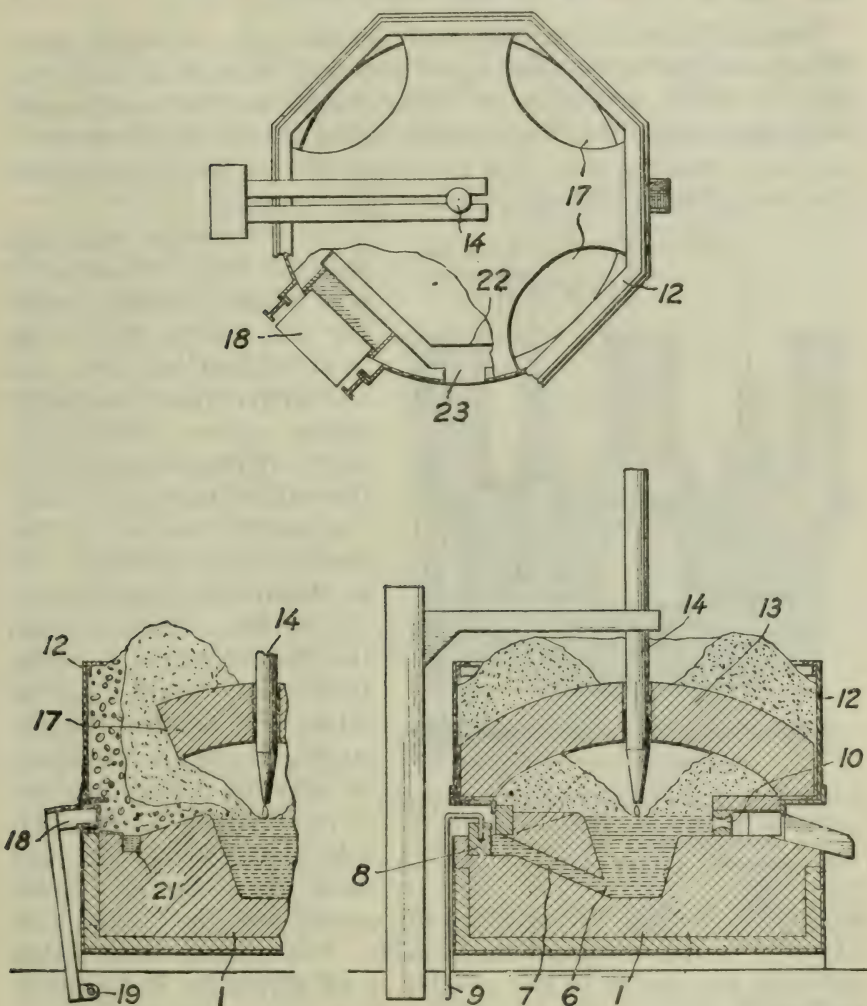


FIGURE 23.—Snyder's Chicago furnace (U. S. Patent 1254079, 1918): 1, Furnace body; 6, bath of lead; 7, passage; 8, block; 9, bus bar; 10, tap hole; 12, frame; 13, roof; 14, upper electrode; 17, charging ports; 18, reciprocating feeder; 21, troughs; 22, collecting trough; 23, outlet.

nace. Mr. Snyder states that these tests were successful, but that after a short run the furnace was converted to other uses and the zinc work dropped. He states the power consumption at about 900 kw. h. per ton of ore, the furnace capacity being about 35 tons a day. The electrode consumption was about 10 pounds per ton of ore, and

furnace labor six-man shifts per 35 tons. No exact data as to metal recoveries are available, but the slags produced were low in metal values.

LOUVRIER FURNACES.

François Louvrier has suggested several types of electric shaft furnace adapted to the smelting of zinc ores. One of his early furnaces is shown in Figure 24. The charge was fed continuously down the shafts *a* into the smelting region around the electrodes *b*, slag being removed at *c*. The zinc vapor was led into the condenser *d* between the two smelting chambers and condensed.

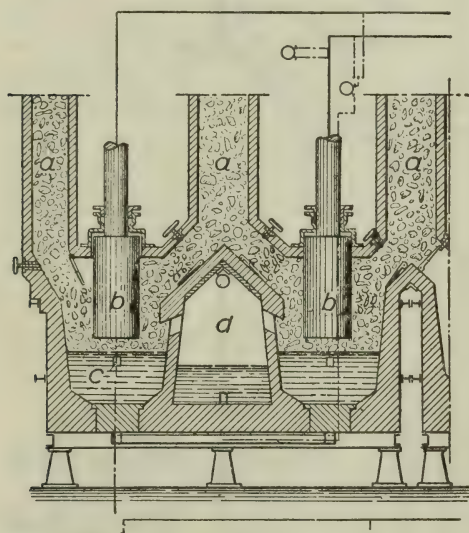


FIGURE 24.—Early Louvrier furnace* (U. S. Patent 989169, Apr. 11, 1911): *a*, Shafts; *b*, electrodes; *c*, opening for removal of slag; *d*, condenser.

Later Louvrier concluded that the large formation of blue powder in electric smelting was due to the intense localized heat employed, and altered his furnace to a single boshed column, with a plurality of electrodes entering the walls of the furnace bosh.

A recent form of his furnace is shown in Figure 25. In this the smelting chamber 2 is of larger diameter than the charge column 3. Electrodes, 12, which can be raised and lowered through stuffing boxes, are introduced at intervals through the top of this smelting chamber outside the feed column, and other electrodes are shown

at 8 and 9, those at 9 being always in contact with the annular ring of metal in the bottom of the crucible. The heating current thus traverses the charge in the crucible in all directions and heats it uniformly.

The upper column of the furnace is double walled, the two walls forming the annular chamber 15. This is provided with a series of shelves 16. Powdered coal is fed through this chamber from the supply bin 17 into the smelting chamber, some of it being always retained on the shelves provided. The zinc vapors produced in smelting are led off over this powdered coal and out through the flue 19 to the condenser. Powdered coal can also be fed around the vertical electrodes, thus protecting them somewhat from wear and oxidation.

THE QUENEAU FURNACE.

During the several years beginning about 1908, A. L. J. Queneau worked with various forms of a rotary electrical furnace, one modification of which is presented in Figure 26. The external steel shell *e* was provided with rings *d*, which rested upon the flanged rollers *g*, and with a band gear *h*, driven by a variable-speed motor so that the whole furnace could be rotated or oscillated. The shell was closed at one end by the metal plate *a'* having the hub projection *s* through which the charge was introduced. This end plate was so fastened to the shell as to be electrically insulated from it and made contact with one lead of the external electric circuit by means of the brush *n*.

The opposite end of the shell was closed with the plate *a*, with a recess, *g*, for cooling with air or water spray. This plate was electrically connected by the flexible connection *b* to the cylindrical shell and thence by the brush *m* to the other lead of the electric circuit. The shell was lined first with a layer of fire brick and inside of that with a layer of magnesite or chrome brick resistant to the slag produced in the furnace.

The end courses *n* were made of blocks of a conducting mixture of graphite, dead burned magnesite, and a carbonaceous binder, baked to drive off hydrocarbons. These were backed up by the graphite end linings *o*.

The conductor that carried the current inside the furnace was a layer of molten iron alloyed with about 12 per cent of phosphorus. This alloy has a higher resistance than pure iron and remains fairly constant in composition. If desired, the slag formed in the furnace could be used alone as resistor, in which case, because of its higher resistance, a higher voltage and smaller current could be used.

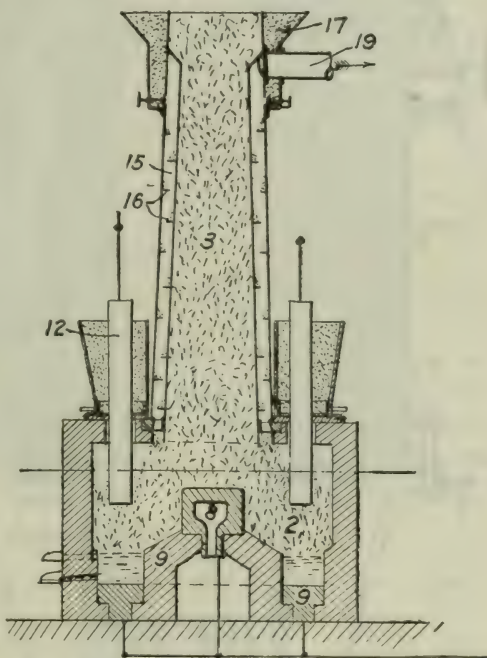


FIGURE 25.—Louvrier furnace of 1920 (U. S. Patent 1342636, June 8, 1920): 2, Smelting chamber; 3, charge column; 8, 9, 12, electrodes; 15, annular chamber; 16, series of shelves; 17, supply bin for powdered coal; 19, flue.

The current thus passed from the lead *r* to the plate *a'*, thence through the graphite *o* and conducting courses *n* to the fluid conductor *t*; thence through the courses *n*, and graphite *o*, at the opposite end to the plate *a* and shell *e* to the opposite lead *m*.

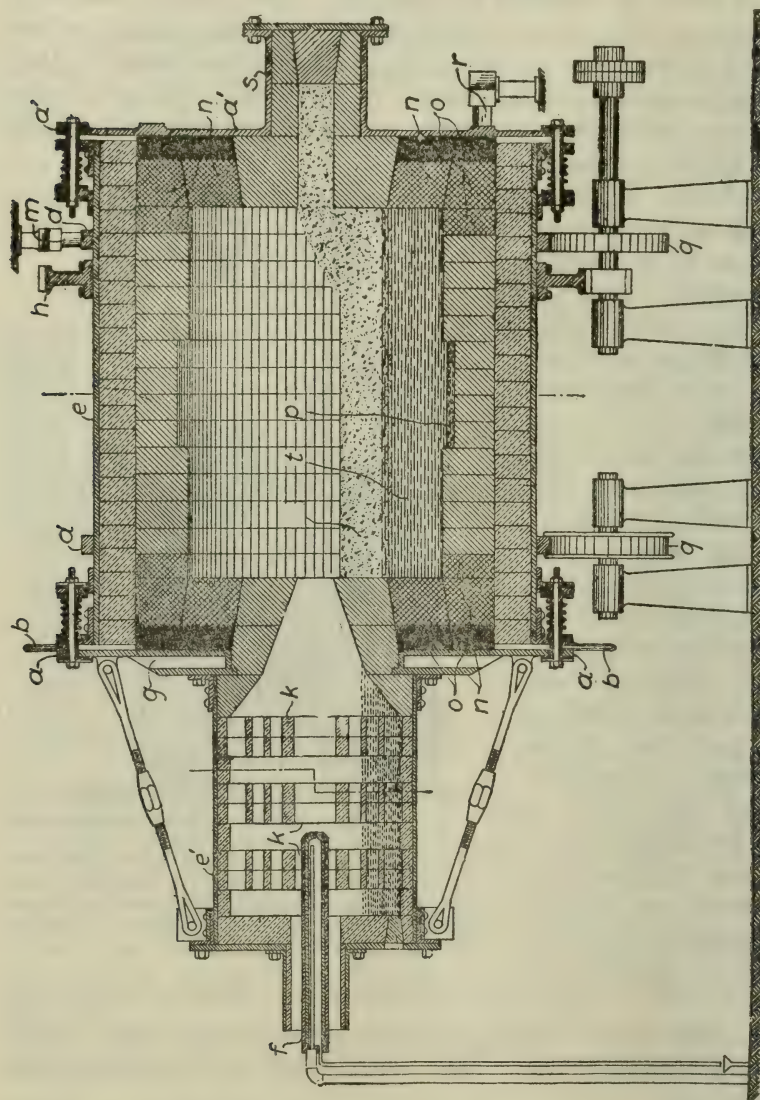


FIGURE 26.—The Queneau rotary furnace (U. S. Patent 1006877, Oct. 24, 1911): *a, a'*, End plate; *b*, flexible connection; *d*, rings; *e*, steel shell; *e'*, condenser shell; *f*, air or water cooling device; *g*, cooling recess; *h*, band gear; *k*, partitions; *m*, brush; *n*, conducting end courses; *o*, graphite and linings; *p*, well for heavy metals; *q*, flanged rollers; *r*, brush; *s*, hub projection; *t*, fluid conductor.

At *p* the diameter of the furnace chamber was increased somewhat to provide a well for any heavy metal contained in the charge.

The condenser shell *e'* was bolted to the end plate of the furnace shell. It was lined with refractory brick and was provided with the perforated partitions *k* to provide condenser surface, and with the air or water cooling device *f*.

Queneau also designed a stationary furnace of the buried-arc or slag-resistance type, which was charged at the side, on the same principle as a copper reverberatory smelting furnace. In this furnace the condenser consisted of a member of short tubes embedded in the end walls of the furnace.

ELECTROTHERMIC ZINC SMELTING EXPERIMENTS OF THE CANADIAN GOVERNMENT.

PURPOSE OF THE INVESTIGATION.

After the closing down of the plant of the Canada Zinc Co. at Nelson, British Columbia, the zinc producers of East and West Kootenay, British Columbia, appealed to the Canadian Government to conduct an investigation for the purpose of developing some process by which the zinc ores of that region could be profitably treated. This question of the development of the zinc industry of British Columbia had been a much agitated question for several years, as the distance to a market for zinc ores made the profitable mining of such ores in the Province difficult.

In response to this appeal, \$50,000 was voted in 1910 by the Dominion Government for the purpose asked. W. R. Ingalls was placed in charge of the investigation and was instructed "to investigate and carry through an investigation for the discovery or development of some method for the economical treatment of the mixed zinc-sulphide ores of Canada, in the production of metallic zinc or a marketable zinc product."

EXPERIMENTS AT MCGILL UNIVERSITY.³²

After a careful survey of the field it was decided that electric smelting offered the most promise, and laboratory experiments were inaugurated at McGill University, first under the immediate direction of Dr. Alfred Stansfield and later of Edward Dedolph. It was realized from the beginning that if electric smelting were to be successful with power costs as in Canada, a large yield of spelter must be obtained in the first smelting with only a small amount of blue powder to be resmelted. To prove that zinc vapor produced in an electric furnace can be successfully condensed to liquid metal under favorable conditions the first experiments consisted of simply volatilizing spelter by electrical heat and recondensing it. Then a series of tests were made with several types of furnace reducing fairly pure zinc oxide. Results from these tests proving favorable, work on roasted zinc concentrates was started.

A prolonged series of small-scale experiments was then carried out during the years 1911 and 1912. Many designs of furnace were

³² Most of the data on experiments at McGill and Nelson are taken from W. R. Ingalls' unpublished report to the Canadian Department of Mines.

used, with various methods of charging, arrangements of electrodes, and condensation apparatus. The detrimental effect of CO_2 was realized and for some time an incandescent coke column between the furnace and condenser was considered necessary. This was, however, subject to clogging and the temperature difficult to regulate, and was later found to be unnecessary if the ore charge were preheated before being charged to the smelting furnace.

The detrimental effect of dust carried into the condenser from the furnace was for a long time not realized in its full importance, but after steps were taken to diminish dusting the condensation results were much improved.

The type of furnace finally decided upon in the small-scale experiments was of the slag-resistance type. The charge was fed through an open shaft and rested upon the slag bath. A brick wall separated the charge shaft from the vaporizing part of the furnace, affording conditions that minimized the dusting of the charge. A furnace of this type, having a capacity of 200 to 250 pounds per 24 hours, gave in a series of runs very encouraging results and it was decided to build a larger furnace at Nelson, British Columbia.

EXPERIMENTS AT NELSON, BRITISH COLUMBIA.

During 1913 a furnace, with a preheater and other accessories, designed for a capacity of 2,000 pounds per 24 hours, was erected at Nelson, in the old plant of the Canada Zinc Co. In this plant the charge of roasted ore and reducer was to be preheated in retorts, measuring 7 by 11 by 62 inches inside dimensions, for 4 hours. Eight of these retorts were sufficient to preheat 2,240 pounds per 24 hours. The preheated charge was pulled from these retorts into heavy pots and transferred to the smelting furnace.

The furnace was of the final type used in the small experiments, as above described, but was so planned that details of construction, such as the charge-feeding device and electrode arrangement could be easily altered if further experiments proved such alteration necessary. The condenser consisted of five common wrought-iron pipes, 5 inches in diameter, heated by a grate fire.

In the first test of this furnace the masonry walls were quickly destroyed, and it was decided that no kind of masonry would be likely to endure the severe conditions of electric zinc smelting. Accordingly a furnace with water-jacketed walls was built. Thenceforth the work at Nelson consisted of efforts to work out a satisfactory furnace construction and method of operation. These efforts were rewarded by a gradual improvement in results, but the original appropriation was used up before the metallurgical difficulties were entirely overcome.

Ingalls, as a result of the work at Nelson, advised that a 1-ton furnace could never be a commercial success, and that it was im-

possible to estimate the cost of developing a larger furnace and overcoming the difficulties that he expected to encounter in going from a 1-ton furnace to one of perhaps 10 tons capacity per day. Furthermore, W. McA. Johnson, who had been working along the same lines, was as far or farther advanced toward a successful process, and was planning to build a 10-ton plant; therefore, Ingalls advised that the work be dropped pending Johnson's further experiments.

PETERSON'S EXPERIMENTS.

During 1912 and 1913, some interesting work was done by Peter E. Peterson at Butte, Mont., on behalf of the Butte & Superior Co.,

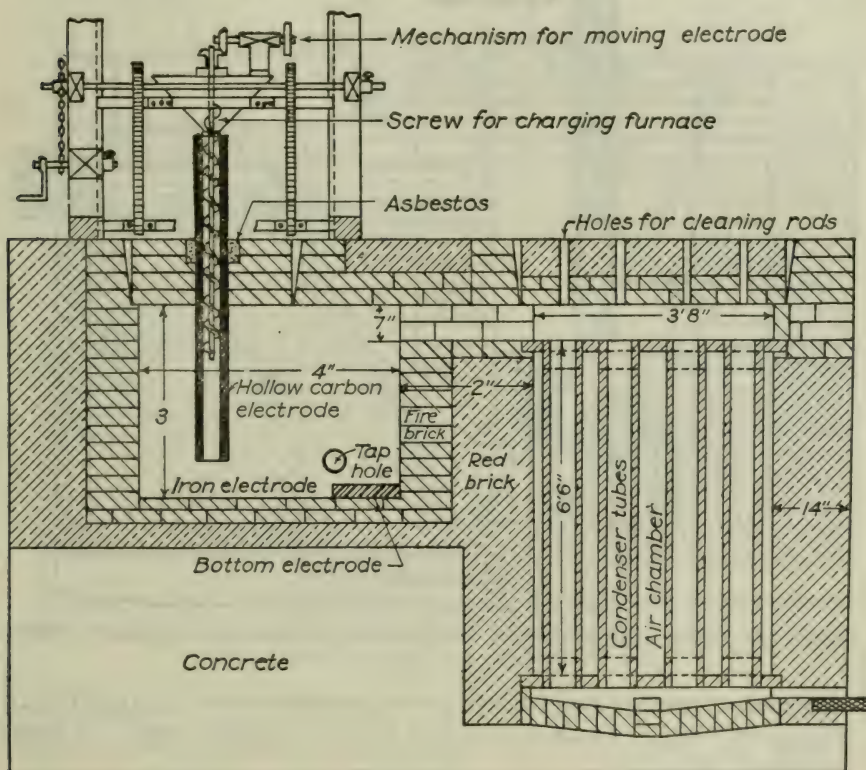


FIGURE 27.—Peterson furnace with hollow electrode (Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 220).

using the buried-arc or slag-resistance type of furnace similar to the Trollhattan and Johnson furnaces, but with novelties in the way of charge feeding devices and condensers.³³

In the first experiments, the precipitation method of treating raw blende with iron or lime and carbon was used. Later this was abandoned in favor of the usual reduction of roasted ore with carbon.

³³ Peterson, P. E., The electric furnace for zinc smelting: Trans. Am. Electrochem. Soc., vol. 24, 1913, pp. 215-239.

One of the earlier furnaces used in the larger-scale experiments is shown in Fig. 27. This furnace was designed for a capacity of 1 ton per 24 hours. The bottom electrode was an iron plate or graphite rod, and the vertical one was a hollow carbon or graphite tube. The charge was introduced into the furnace through this hollow electrode by a screw feeding device. This furnace gave a low power consumption—1,100 to 1,300 kw. h. per ton of 50 per cent zinc concentrate—but was mechanically unsatisfactory.

The condenser consisted of eight clay tubes surrounded by cold-air passages. Another condenser used with this type of furnace

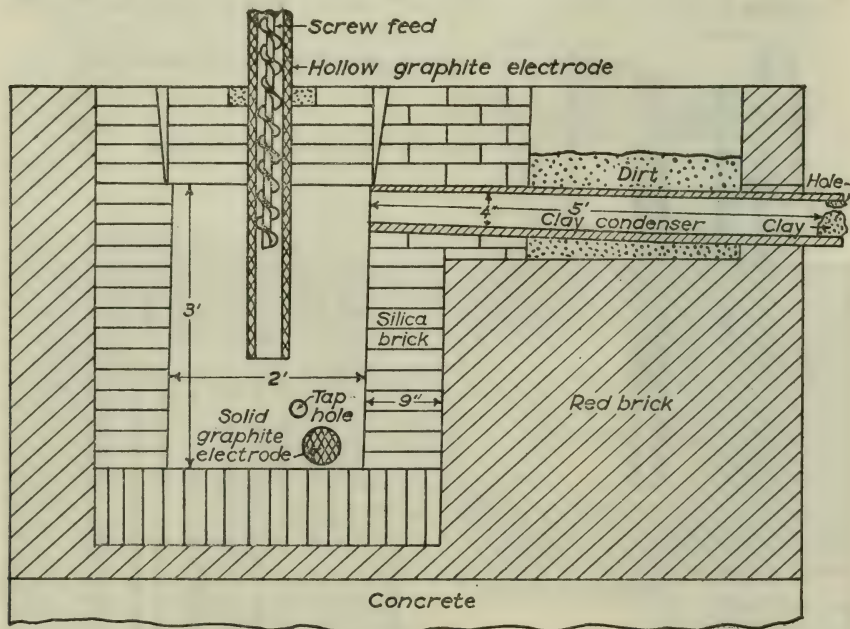


FIGURE 28.—Hollow electrode furnace with multiple horizontal-tube condenser (Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 225).

consisted of several horizontal clay or graphite tubes, arranged as in Figure 28, which could be covered with a variable depth of heat-insulating material.

The first condenser could not be kept hot enough for good condensation; it consequently produced mostly blue powder and was decided to be too complicated for practical use. The horizontal tubes in the second condenser gave some spelter but did not have capacity enough to condense all the zinc vapor produced. A later run was made with this latter condenser, in which the furnace was run on barren material until the condenser reached 859° C. The zinc charge was then introduced and fed at such a slow rate that very little zinc escaped from the end of the condenser tubes. Dur-

ing 12 hours of smelting in this way, 86.4 per cent of the zinc charged to the furnace was recovered as liquid zinc, but the power consumption was 4,000 kw. h. per ton of 50 per cent zinc concentrate. The principle of this condenser was evidently good, but its capacity was not proportioned properly to that of the furnace.

It was decided that the horizontal bottom electrode was unsatisfactory, so it was given up and a furnace using two nearly vertical electrodes was tried. One of these furnaces is shown in Figure 29.

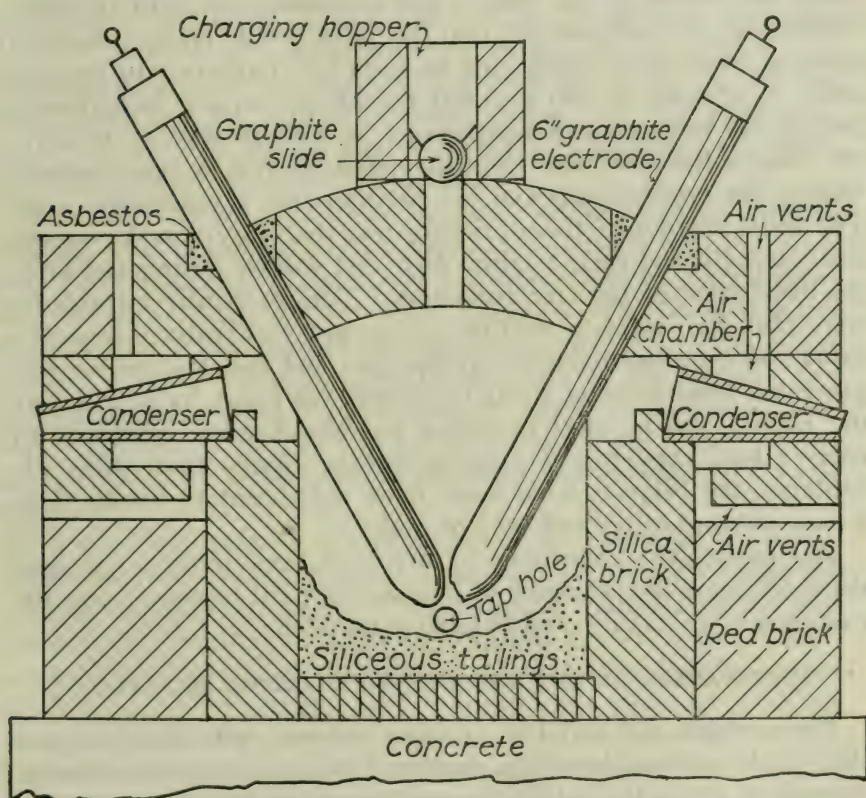


FIGURE 29.—Peterson buried-arc furnace (Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 225).

The smelting operation in this furnace gave no trouble during an actual running time of 30 days, but the same condensation difficulties arose.

In mixing the charge, the plan was to produce slags having such a high formation temperature that the zinc would be volatilized before slagging of the charge began. It was found possible to reduce the zinc content of the slag to any desired figure. Coke was used as a reducer. Sufficient sulphur was left in the roasted concentrates and enough copper ore was added to the charge to pro-

duce some matte to serve as a collector for the precious metals. The company decided that the work was not of commercial promise and abandoned it in 1913.

FULTON ELECTRO-RESISTANCE BRIQUET FURNACE.

THE INDESTRUCTIBLE BRIQUET.

In 1914, Charles H. Fulton proposed,³⁴ as an improvement in the retort distillation of zinc ores, to mix the roasted ore with an excess of fine coke and sufficient hot coal tar, pitch or similar carbonaceous material to act as a binder, and to form this mixture into briquets under a pressure of 500 to 1,000 pounds per square inch, finally baking the briquets at a temperature of 450° to 500° C. to drive off volatile hydrocarbons from the pitch, leaving a coke network uniting the original coke and ore particles into a continuous mass. The novel feature of these briquets was that they would preserve their strength and their original form and volume during and after the distillation of the zinc from them. The particular advantages were that the intimate mixture of the ore with carbon caused a rapid and complete reduction of the zinc, and that the residue was prevented from forming a slag to corrode the retort walls, and that the residue was in ideal condition for blast-furnace treatment to recover lead, copper, and precious metals. Aside from these advantages, the briquets were much more convenient to handle than a loose mixture of ore and coke or coal.

The particular proportions of ore, coke, and pitch to be used depended upon the character of the ore, a typical mixture being 100 parts ore, 60 parts coke, and 18 to 20 parts pitch.

APPLICATION OF THE BRIQUET TO ELECTRIC SMELTING.

The strength and stability of these briquets, with the fact that they would conduct electricity, suggested the feasibility of an electric furnace in which the baked briquets themselves would serve as the resistor.

Experimental work with such a furnace was begun in Cleveland, Ohio, in 1914, and in 1916 a plant was erected in East St. Louis, Ill., for the purpose of working out the technical details of the process on a commercial scale. Experiments were continued at this plant, with very promising results, until January, 1918, when the unsettled labor and material markets, due to the war, forced the suspension of the work.

³⁴ Fulton, C. H., Recovery of zinc, U. S. Patent 1193680, Aug. 8, 1916.

EXPERIMENTS AT EAST ST. LOUIS.

The work at East St. Louis was very completely discussed by Fulton in a paper read before the American Institute of Mining and Metallurgical Engineers in 1919.³⁵ Most of the following description is taken from this paper.

BRIQUETS.

The briquets used in most of the work at East St. Louis were cylindrical, 9.25 inches in diameter and 21 inches long, weighing about 90 pounds, and containing 50 pounds of ore. They were formed by mixing the coke and ore, then heating and mixing with the melted pitch in a pug mill and pressing. The mold had a temperature of 75° to 90° C. for the best results and the pressure used varied from 500 to 1,000 pounds per square inch.

The size of the ore used in the briquets, within reasonable limits, has little effect either on the strength of the briquet or on the extraction of the zinc; ore which passes a 200-mesh screen gives practically the same result as that which passes a 10-mesh screen and 86 per cent of which remains on a 35-mesh screen. The size of the coke has a strong influence upon the strength of the briquet, very fine coke giving a much stronger briquet than comparatively coarse coke. With very fine coke and ore, however, it is necessary to use more of the pitch binder, thus increasing the cost of the briquet somewhat.

The coke used must be enough to reduce all the reducible oxides of the ore and leave an amount sufficient to give the briquet the requisite stability and electrical conductivity. Ordinarily the briquet after the distillation of the zinc should have at least 40 to 50 per cent of the weight of the original briquet, depending somewhat upon the nature of the residue. When the ore gives an infusible granular residue the briquet has more strength after distillation than when this residue tends to fuse into globules. An ore that leaves a residue of reduced metal has a higher electrical conductivity than the average, whereas an ore with a lime gangue may form calcium carbide, thus giving the briquet a high resistance. From 50 to 75 per cent of coke is usually required. This is more than is used in the retort process, but the distilled briquets can be crushed and used a second time as coke, or may be used as a high-ash fuel.

The usual proportion of pitch required is 10 to 15 per cent, more being required for fine ore and coke than for coarse. More pitch than this would be squeezed out in pressing the briquet, while less than this does not give a satisfactory briquet. A high-melting

³⁵ Fulton, C. H., Electric resistance furnace of large capacity for zinc ores: Trans. Am. Inst. Min. and Met. Eng., vol. 64, 1920, p. 188.

pitch (melting point 170° to 200° F.) giving upon distillation 55 to 60 per cent of good, firm coke is the best binder. The pitch may be partly replaced by coking coal, but with less satisfactory results.

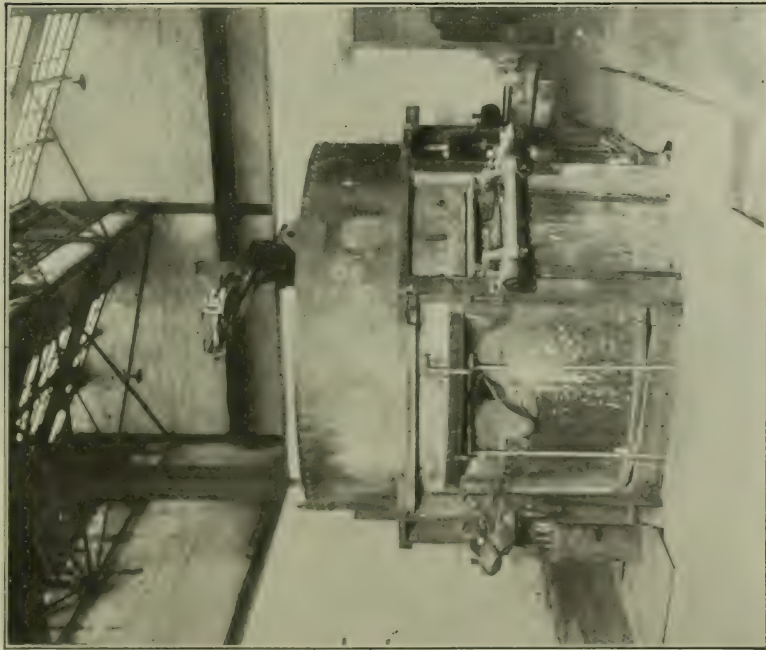
After each briquet was pressed it was set on end in a light sheet-iron cylinder slightly larger than the briquet on a flat car holding 20 briquets, and the space between the briquet and cylinder filled with crushed coke to prevent the oxidation of the surface of the briquet during baking. The car was then pushed into the baking oven. The baking was continued for 6 to 8 hours with a final temperature of 450° to 500° C. for the last few hours. This baking served to coke the pitch binder and after baking the briquet could be thrown violently to the floor without breaking. During the baking the briquet passes through a soft stage up to about 300° C. and above this temperature gradually increases in strength as the distillation of the pitch continues.

The electrical resistivity of the raw briquet is 26 to 30 ohms per cubic inch. This drops slowly as the pitch is coked until at 450° to 500° C. it drops suddenly to 0.6 to 0.7 ohm per cubic inch. At 900° to $1,100^{\circ}$ C. it is 0.015 to 0.04 ohm per cubic inch.

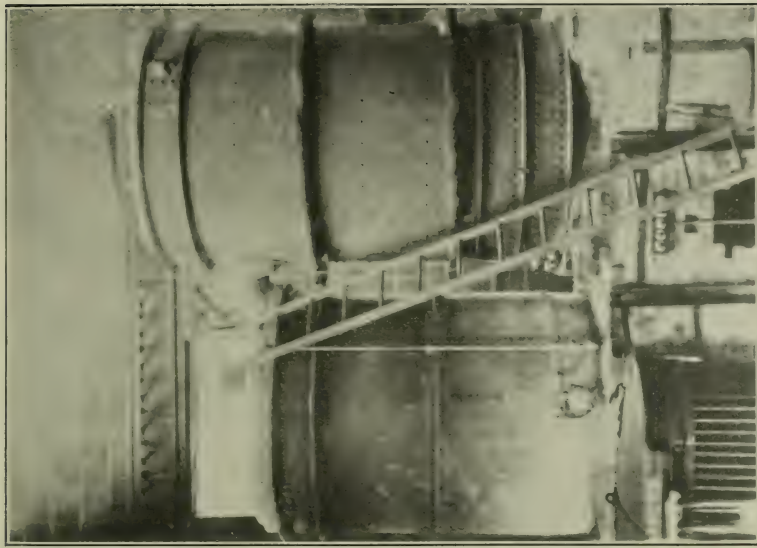
FURNACES.

The furnaces used at East St. Louis held a charge of 36 briquets, arranged in 12 columns of three each, set within a circle and operated on a three-phase circuit, four columns to a phase, connected in the usual Y connection. The amount of ore in one charge in this furnace was approximately 1,700 pounds, requiring 6 hours for distillation. Two hours were required for setting up the charge and discharging, so that three charges could be worked off per 24 hours, giving a daily capacity of 5,100 pounds of zinc concentrate.

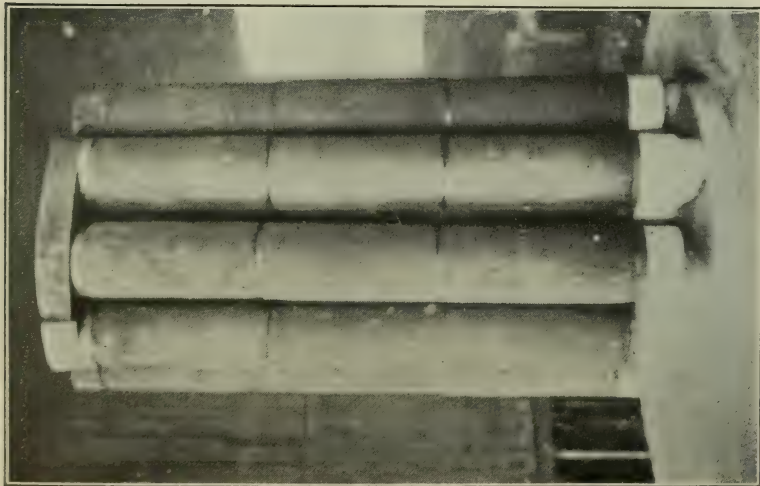
One of the early furnaces, designated as furnace D, is shown in Figure 30. Plate I, *B*, is a photograph of the same furnace. The condenser structure rested upon the center part of a specially constructed, three-part base and the end parts contained the electrodes and supported the briquet charge and covering retort. In operation, the briquet charge was set up on one retort base, the retort lowered into place, and the condenser placed in position. The joint between retort and condenser was made tight with a luting of clay. The current was then turned on and the charge distilled. Meanwhile a charge was set up on the other retort base. As soon as the first charge was finished the condenser was lifted out of the way with the crane. The retort was then lifted about 3 inches and moved to one side, sweeping the briquets over an apron plate to the floor, leaving the base ready for setting up a new charge. The hot retort was at once transferred to the other base, the condenser turned around and



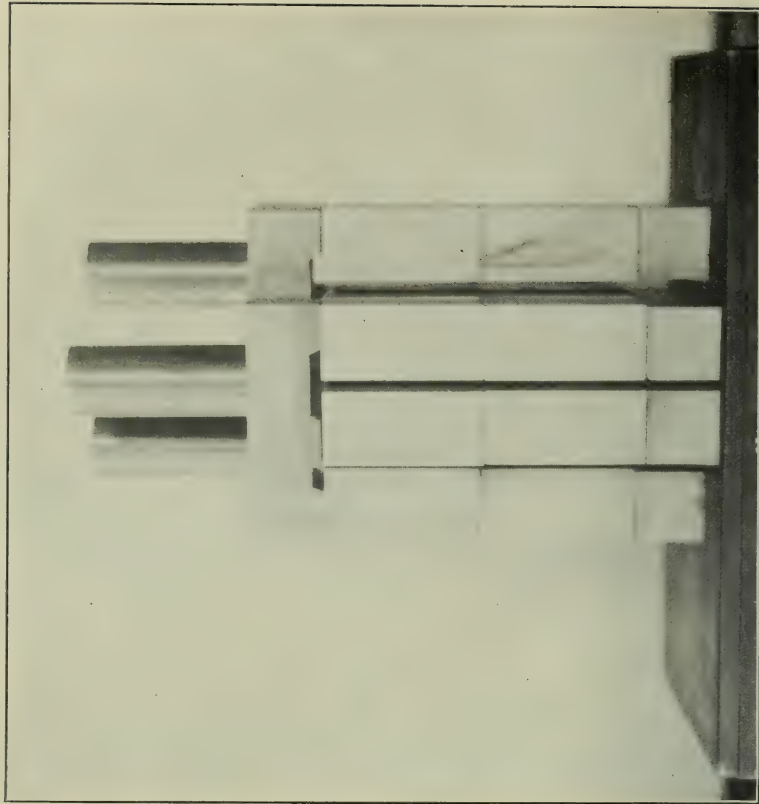
1. SNYDER'S FURNACE AT CHICAGO.



B. FULTON'S EAST ST. LOUIS FURNACE



A. BRIQUET CHARGE FOR FULTON'S EAST
ST. LOUIS FURNACE.



B. SET-UP CHARGE; SQUARE BRIQUETS.

lowered into place, and the second charge distilled. The two end bases were thus used alternately, so that the retort and condenser were never allowed to cool, and the same condenser served for both retort bases. When a cold condenser was started up, it could be heated by an oil burner introduced into the top of the large compartment, allowing the products of combustion to pass under the partition up the smaller compartment and out of the connector flue. With continuous operation, radiation about balanced the heat input due to the condensation of the zinc vapor and the condenser maintained

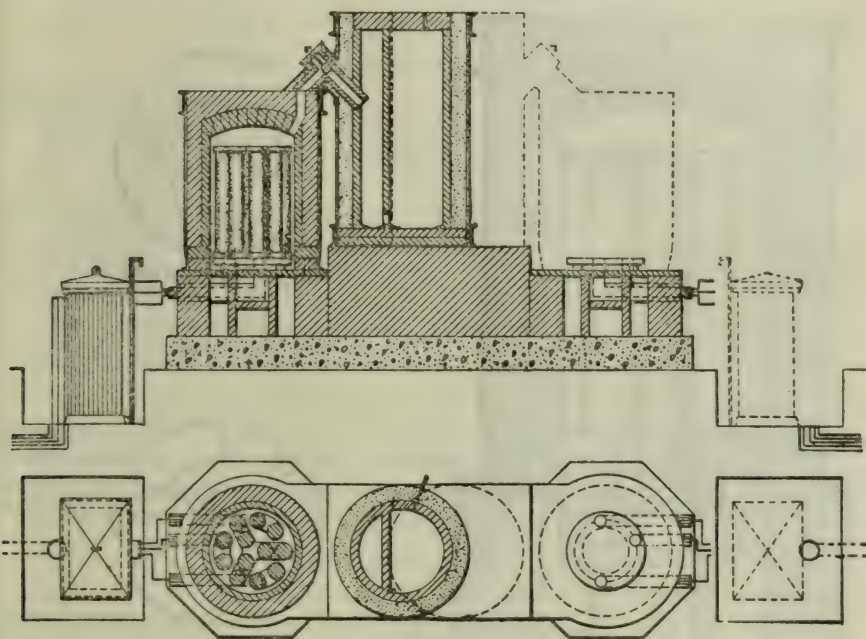


FIGURE 30.—Fulton furnace D.

itself at a temperature of about 700° C. at the inlet and 500° C. at the exit.

The retort consisted of a steel shell with a refractory lining. The inner lining was of high-grade fire brick and the space between this lining and the shell was filled with sil-o-cel for heat insulation.

The electrodes, which were of graphite, 6 inches square, entered the base of the furnace. On these were set 6 or 9 inch risers which reached up through holes in the base. The space between the risers and the walls of the opening was packed with finely crushed carbon. Graphite blocks rested upon these vertical plugs, and upon them were set the columns of briquets. The top connections were also made with 5-inch graphite blocks. Figure 31 shows the electrical arrangement and Plate II, A, is a photograph of the complete set-up.

A charge could be set up in from two to three hours by two men. After the charge was set up, a fire-clay luting was spread on the base where it made contact with the retort, and the latter was lowered into position by the crane.

With hot retorts, the voltage supplied to this furnace at the beginning of a charge was about 150 volts with an energy input of 120 to 150 kw., dropping to 75 to 90 volts with an energy input of 122 to 184 kw. as the distillation proceeded. This was the limit of the electrical equipment, and later with electrical equipment of

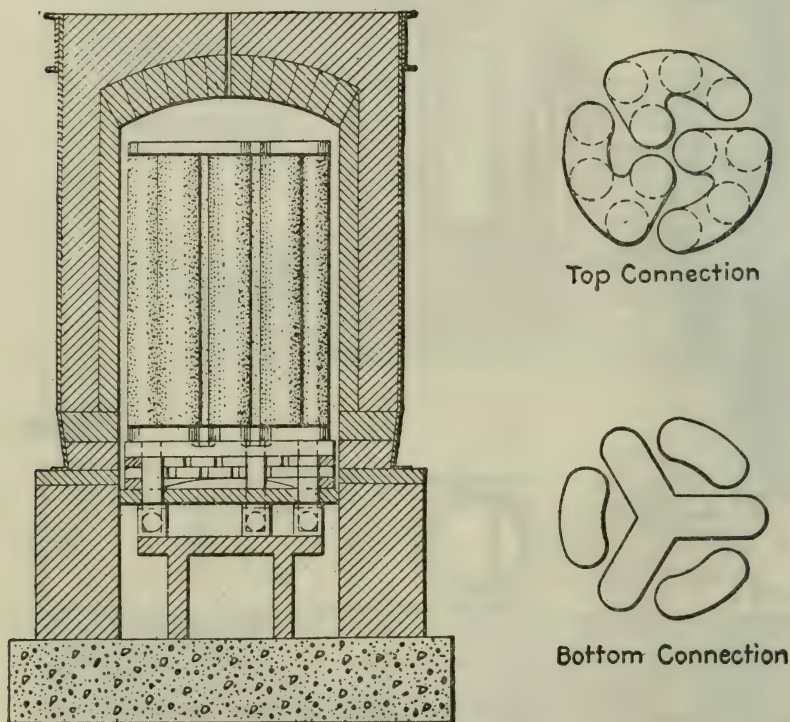


FIGURE 31.—Electrical connections of Fulton's East St. Louis furnace.

larger capacity it was shown that the furnace could easily take 200 to 250 kw. With the low-power input into furnaces D and E, the time of distillation was over eight hours and the power consumption greater, due to greater radiation, than that obtained later with a higher power input. The power factor was about 90.

Tables 2 and 3 show metal balances for several runs with this furnace. The furnace was practically new at the beginning of each series of runs, so that absorption of zinc was high.

Furnace E was a variation of furnace D and results were about the same. About 60 per cent of the zinc present in the charges was recovered as cast spelter, over 1,100 pounds being drawn from the condenser in one tap.

Furnace F was designed for use with new electrical equipment of larger capacity. There were three adjoining retort bases, a movable condenser, and one of the old retort bases arranged to preheat retorts by means of oil burners. The bases proper, which rested on the furnace foundation, were so built that they were transferable; extra ones were supplied, hence the base could be set on a car, the raw briquets placed on it, and the whole set-up moved into the baking oven and baked, and then transferred while still hot to the distilling furnace.

TABLE 2.—*Metal balance of Fulton furnace.*

[Runs 7, 8, 9, 10, and 11, May, 1917.]

Zinc in baked briquets:	Pounds.
Each charge	573.0
Zinc in five charges, total charged.....	2,855.0
Zinc remaining in distilled briquets:	
Run 1, 2.36 per cent.....	52.4
Run 2, 0.67 per cent.....	14.0
Run 3, 1.80 per cent.....	39.0
Run 4, 0.93 per cent.....	19.4
Run 5, 5.97 per cent.....	151.0
Total zinc in distilled briquets.....	275.8
Zinc cast as spelter.....	1,242.5
Zinc in scrapings from bridge wall, rechargeable at 41.75 per cent zinc..	899.0
Zinc in brick from condenser, at 1.74 per cent zinc.....	115.3
Zinc unaccounted for.....	322.4

SUMMARY.

	Per cent.	Pounds.
Spelter tapped.....	42.7	1,242.5
Scrapings, rechargeable.....	30.8	899.0
Zinc in brick from condenser.....	3.9	115.3
Zinc in distilled briquets.....	^a 9.1	275.8
Unaccounted for	^b 13.5	322.4
	100.0	2,855.0

^a The amount of zinc left in briquets is entirely at the will of the operator; this is shown on the individual runs. The runs in this series were, in part, undertaken to obtain figures on power consumption, and certain runs were discontinued at given power inputs to determine proper point of stopping.

^b This covers zinc absorption in the retort hood, base, and passageways and includes zinc escaping uncondensed.

TABLE 3.—*Metal balance of Fulton furnace.*

[Runs 12, 13, and 14, May 9-11, 1917.]

Zinc in baked briquets:	Pounds.
Each charge.....	573.0
Total zinc in three charges.....	1,719.0

Zinc remaining in distilled briquets:

Run 12, 6.57 per cent.....	169.0
Run 13.....	Nil.
Run 14, 0.131 per cent.....	2.6
Total zinc in distilled briquets.....	171.6
Zinc in scrapings from condenser, rechargeable, at 60.85 per cent zinc..	496.0
Zinc cast as spelter.....	903.5
Zinc unaccounted for.....	147.9

SUMMARY.

	Per cent.	Pounds.
Spelter tapped.....	52.60	903.5
Scrapings, rechargeable.....	28.80	496.0
Zinc left in distilled briquets.....	^a 9.83	171.6
Unaccounted for.....	^b 8.77	147.6
	100.00	1,719.0

^a The amount of zinc left in briquets is entirely at the will of the operator; this is shown in the individual runs. The runs in this series were in part undertaken to obtain figures on power consumption, and certain runs were discontinued at given power inputs to determine proper point of stopping.

^b This covers zinc absorption in retort hood, base, and passageway, and condenser brick-work, and includes zinc escaping uncondensed.

The charge of briquets was then further heated by placing the hot retort from the previous charge over the new charge and allowing it to communicate part of its heat to the briquets. After about 1 hour, it was replaced with another retort, which had been preheated to about 1,500° C. with oil. After this had been on about 1 hour and the charge had reached distilling temperature, the current was turned on and the distillation finished by electric heat. To the cost of power must be added the cost of the oil for preheating the retorts, but the electrical power consumption was considerably reduced.

POWER CONSUMPTION.

Table 4 gives power-consumption figures for a series of runs. The initial charges with cold retorts and some underdistilled and overdistilled charges are omitted, as the power consumption in these cases varies from the normal. Runs 25 to 30 were with preheated retorts as described under furnace F.

TABLE 4.—*Power consumption at East St. Louis.*

Run number.	Kw. h. per ton of ore.	Per cent zinc distilled from charge, ^a
2	2, 270	100. 0
5	1, 645	98. 0
8	2, 240	98. 0
9	2, 180	93. 0
16	2, 200	96. 0
18	2, 120	98. 0
20	1, 760	91. 0
21	1, 720	95. 0
22	1, 920	97. 0
23	1, 760	95. 0
25	1, 665	99. 0
27	1, 548	100. 0
28 ^b	1, 365	99. 0
29	1, 811	99. 0
30	1, 237	99. 0

^a Where less than all the zinc is distilled, the operation was purposely stopped to determine relation of extraction and power consumption.

^b Franklinite charge.

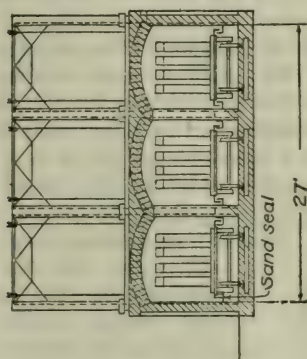
The theoretical power consumption for a 60 per cent zinc ore made into briquets of the usual composition was calculated to be 1,372 kw. h. exclusive of radiation losses. Of this, 421 kw. h. required to preheat the charges to 920° C., could be supplied by some such method as preheated retorts; and the carbon monoxide produced in the distillation is equivalent to 154 kw. h.

The energy in the briquet residue in cooling from 1,200° to 25° C. is 378 kw. h. Radiation can be made very small, as the heat is developed in the briquets themselves.

Runs 28, 29, and 30 were made to produce zinc dust with furnace F. A good grade of zinc dust, free from oxide, was made, 97 per cent passing through a 200-mesh screen; recoveries were very high.

The particular advantage claimed for the Fulton process is that it combines the physical and metallurgical characteristics of present-day retort practice with the other advantages possible in the electric furnace, such as generation of the heat in the charge itself, large units, low labor cost, and applicability to the treatment of low-grade ores. The charge is distilled in a closed retort within which a uniform temperature is maintained. This produces a gas nearly free from carbon dioxide, from which zinc can be condensed as spelter without the formation of blue powder. The process is applicable to high-grade ores of any composition and to complex ores, such as those of the Butte district. Impurities, such as iron and lead, which are so harmful in the retort process have no deleterious effect in the electric briquet furnace.

material, it is assumed that one of three processes may be used: 1. The standard retort process. 2. The electrolytic process. 3. An electrothermic process. In all three methods the first step is roasting if sulphide concentrate is used.



Therefore in the proposed plant, the roasting furnaces are not considered and the start is made with calcines. Figure 32 is a flow sheet of the proposed plant. It consists of three sections: (1) The coke and briquet bins and crushing section, (2) the mixing and briquetting section, (3) the baking ovens and electric-distilling furnaces. The crushing plant is standard and needs no description. The coke and spent or distilled briquets are crushed to such fineness that the maximum size of particle is approximately one-eighth inch. The calcines and

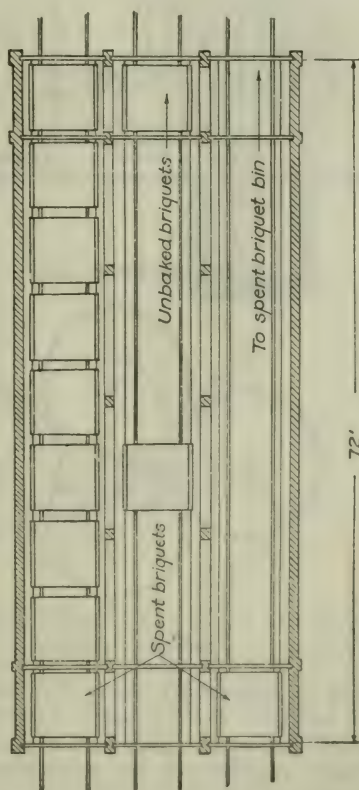
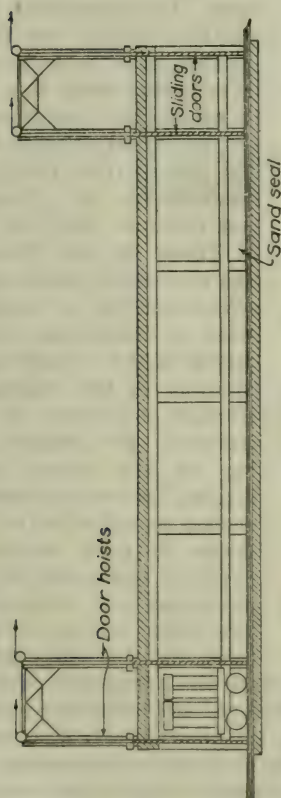


FIGURE 33.—Briquet baking oven.

the coke are fed in the proportion of 100 to 60 by conveyers to a concrete mixer and the mixture elevated to storage bins, from which it passes to the mixing plant which prepares it for briquetting. The mixing of the ore and coke with

the pitch is done in a standard asphalt road plant. In this the ore-coke mixture is heated to approximately 200° C. in a sand drier and raised to the sand bin and thence into a weigh box which in turn discharges into a knife-blade mixer. At this point the 18 to 20 parts of molten pitch are added by means shown in the illustration. The system of mixing results in a complete coating of the ore and coke particles with pitch. The mixer discharges to the hopper of the briquetting process by means of a conveyer. The press is standard and is known as a caking press used in other industries. The large briquets, 12 by 12 inches cross section and 27 inches high, are discharged on to a traveling table conveyer and then set up by hand in charge form on a furnace base placed on a baking oven car. These cars pass into the baking oven which is constructed on the

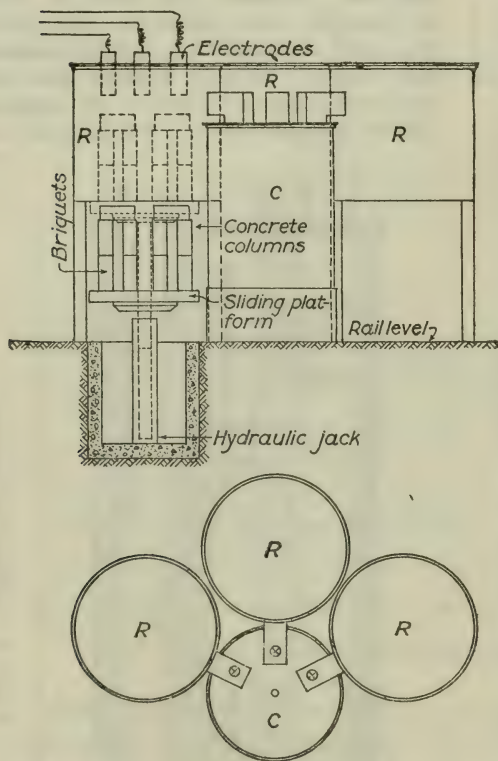


FIGURE 34.—Retort and condenser of plant proposed by Fulton.

recuperative principle and is shown in diagram form in Figure 33. The heat for baking is derived from the hot spent briquet charge which as it comes from the retort is immediately pushed into the baking oven. The distilling retorts, nine in number, are fixed in position and the furnace base is movable. This is a return to an original design used in the earlier experimental work and differs from the East St. Louis design of a fixed base and movable retort as shown in Figure 30 and Plate I, B. The fixed-retort design presents advantages for a commercial plant in connection with the baking of the briquets and the handling of the charge. The graphite electrodes (9 inches diameter) are located in the top of the retort, passing through coke-sealed glands and making contact with the briquet charge by their weight. The electrical equipment is located on a platform approximately level with the top of the retorts and the bus bars carrying the current are short. The furnace base with its charge, as it comes from the baking oven on a car, is run over the hydraulic lift below the retort and raised into place. The base is made tight to the retort by a ring of fire-clay luting. In operating, the furnace base with its distilled or spent charge will be lowered by the lift to a car and at once transferred to the baking oven, while a furnace base with a baked charge is taken from the oven and immediately lifted to the hot retort.

The three retorts in a group discharge into one condenser. The arrangement is shown in Figures 34 and 35. The connections for conducting the

zinc vapor from the retort into the condenser are permanently attached to the condenser and are as short and simple as possible. The condenser can be removed from its position by crane and replaced by another one when necessary.

Figure 36 illustrates the retort and condenser in diagram form and gives dimensions. The lining of the retort is high-grade fire brick and the insulating material is sil-o-cel. The lining of the condenser must be of a very pure fire clay or a synthetic refractory practically free from iron. This freedom from iron is one of the essential requirements of the condenser lining, since

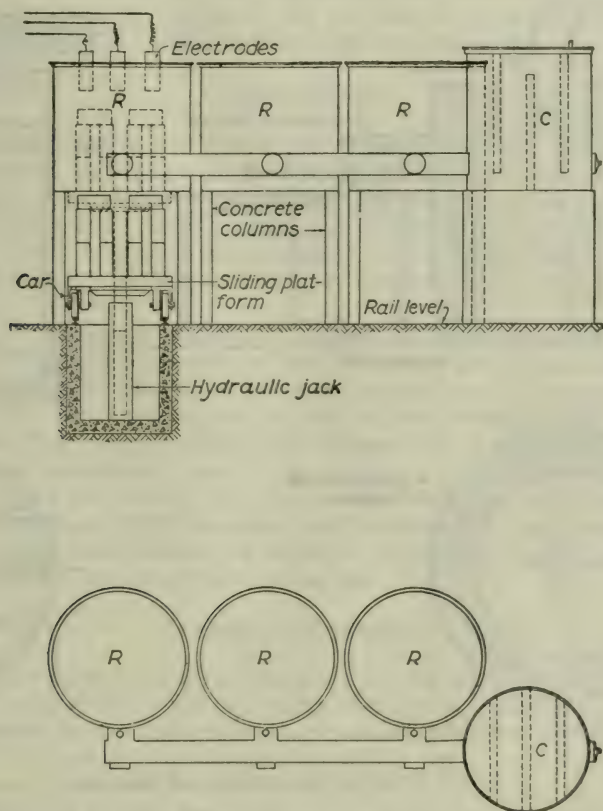


FIGURE 35.—Retort and condenser, alternate plan.

the presence of iron oxide particles in the lining leads to its destruction by carbon deposition from carbon monoxide gas—a reaction very active at about 500°C ., a prevailing temperature in some regions of the condenser. The arrangement of one condenser for three retorts permits of a continuous flow of zinc vapor to the condenser. The condenser may be arranged with either two or four passes or compartments, this arrangement being attained by curtain walls. With the combination of three retorts and one condenser, the volume of the condenser should be not less than twice the volume of the retort. The factors involved in condensation are temperature, volume, and surface. The temperature in the condenser will range from about $1,050^{\circ}\text{C}$. at the intake to about 470°C . at the gas exit. For every pound of zinc condensed from a mixture of equal volumes of zinc vapor and carbon monoxide

between 1,050° and 650° C., there is liberated 1,050 B. t. u., or 1,250,000 B. t. u. per hour for the condenser with three retorts as described. If the radiation from the condenser is greater than the quantity of heat supplied by the condensation of metal, the condenser will have to be heated. On the other hand, if it is less, the condenser will have to be cooled. In the design under discussion, the condenser will have to be cooled, which may be accomplished by water sprays. In starting a new condenser, it will be preheated to the proper temperature (700° C.) by means of an oil burner introduced into a handhole at the bottom of one of the passes, the condenser connection acting

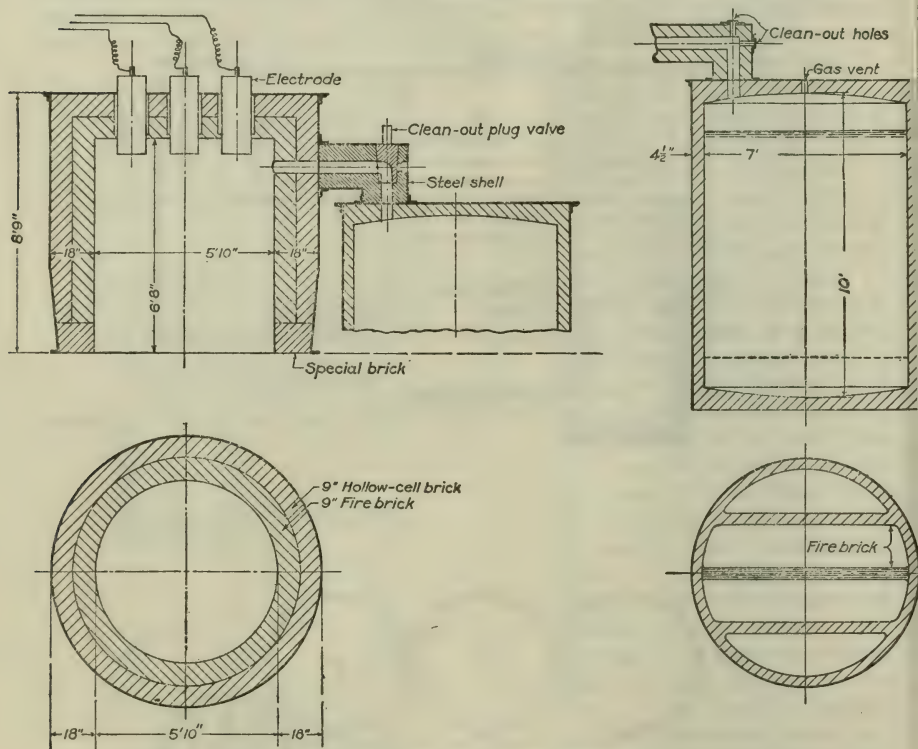


FIGURE 36.—Diagram of retort and condenser.

as a chimney. In condensing from a mixture of equal volumes of CO and Zn vapor theoretically 85 per cent of the zinc may be condensed to liquid metal with an exit temperature of 750°. In practice, it is very probable that two condensers will be placed in tandem, one operating between temperatures of 900° and 750° C. and the other between temperatures of 750° C. and 450° C. This system will afford good control. In the East St. Louis pilot furnace no difficulty was experienced in condensation due to the formation of blue powder. This is a common difficulty with electrothermic zinc smelting, but in this process the metallurgical conditions of the retort process are retained, namely, a uniformly heated space above 1,000° C. and a preheated charge so that no gases are present in the condenser except Zn vapor and carbon monoxide. It is the presence of such gases as CO₂, water vapor, etc., which leads to the formation of blue powder.

The spent briquets contain the residue of the ore. With the usual high-grade zinc concentrate this residue is from 12 to 20 per cent of the weight of the concentrate and from 15 to 25 per cent of the weight of the spent briquet, the rest of this being carbon—that is, the spent briquet is practically a high-ash coke. The briquets will be recrushed and used over again at least once, sometimes twice. If the ore contains lead and silver, approximately 80 to 85 per cent of the lead and over 90 per cent of the silver may be retained in the briquet by the close temperature control of distillation which the process permits. By using spent briquets containing lead and silver over a number of times, a relatively high concentration of these two metals may be obtained. This applies also to copper in complex zinc ores. By the reuse of the briquets the reduction fuel for the process is reduced to 30 per cent and less of the weight of the ore.

THE CHARGE.

The following figures give details for the briquet charge. The briquet mixture is assumed, for example, to be 100 parts calcine (Missouri concentrate, 60 per cent Zn), 60 parts coke, 20 parts pitch. In briquetted form this weighs 115 pounds per cubic foot. During baking approximately one-half of the weight of the pitch is distilled off and baked briquets contain 56 per cent ore, equivalent to 34 per cent zinc. The briquet is 12 by 12 inches in cross section and 27 inches long.

Plate II, *B*, illustrates a charge of briquets set up for introduction into the retort. In this charge, briquets are used as top and bottom connectors, so that the whole charge is made up of briquets. In East St. Louis, the bottom and top connectors were made of graphite, but briquet connectors present manifest advantages. The charge consists of 12 columns^{55b} of 2 each, 6 top connectors, and 6 bottom connectors, a total of 36 briquets. A briquet contains $2\frac{1}{2}$ cubic feet and weighs 260 pounds. The charge contains 81 cubic feet, weighs 9,315 pounds, and contains 5,216 pounds of ore, equivalent to 3,167 pounds of zinc. A retort distills three charges per 24 hours, so that the capacity per retort is 15,648 pounds, or 7.824 tons of calcine per day.

ELECTRICAL EQUIPMENT.

The furnace is operated with three-phase alternating current. The graphite electrodes make contact with the charge and are not consumed. The charge contains approximately 5,400 pounds of concentrate, and on the basis of 1,500 kw. h. per ton and a 6-hour distillation period (these two figures purposely assumed large, to provide proper margin), 4,050 kw. h. will be used in 6 hours, or an intensity of 657 kw. per retort. For the nine retorts, giving a capacity of $(2.7 \text{ by } 3 \text{ by } 9) = 72.9$ tons calcine per day, or 2,187 tons per month of 30 days, or 2,573 tons green concentrate (a margin of 573 tons to allow for delays) there will be required nine 3-phase 60-cycle 700 kv. a. transformers, nine 3-phase 60-cycle 160 kv. a. regulators, and nine complete switchboard equipments. These are designed to give full kv. a. capacities at secondary voltages from 40 to 250 volts, with a primary voltage of 2,300 volts. Other primary voltages can be used equally well. It is assumed that the current can be purchased and is not generated at the plant. Other standard frequencies than 60 may be employed.

The specific resistivity of the briquet ranges from 0.01 to 0.04 ohm per cubic inch, dependent on the nature of the ore; 0.02 to 0.03 ohm is the usual figure.

^{55b} For the best electrical connections it is almost essential to use 12 columns. Other numbers, while possible, present decided disadvantages in the set up.

A charge set up as in Plate II, *B*, shows a standard delta connection in the briquet columns, with four columns in series, and requires a terminal voltage of 217 volts at the electrodes for an intensity of energy input of 675 kilowatts; connections with the charge can, however, be made so as to lower this voltage if it should be desired. The transformers are set close to the retorts on a platform to keep the reactance low. By using electrodes in the top of the retorts the secondary leads of the transformers are about the same elevation as the electrode terminals of the furnaces, and short bus connections are obtained. All high-voltage control and all switchboard equipment will be outside the furnace room.

RECAPITULATION AND ADVANTAGES OF THE PROCESS.

The process differs essentially from other electrothermic processes for zinc. It is not a smelting process but a distillation process like the retort process. Most of the proposed electrothermic processes mix the ore with a limited amount of reduction fuel and smelt by electric heat generated by a buried arc. The furnace space is not uniformly heated and carbon dioxide gas and water vapor are present with the zinc vapor and carbon monoxide and condensation to liquid metal is very difficult and the amount of blue powder formed is large. The residue of the ore is smelted to slag which usually contains considerable zinc. It seems questionable whether in such a furnace it is possible to recover other metals in the form of bullion or matte on account of the high temperature prevailing. In Norway and Sweden electrometallurgists have gone frankly to vaporization of lead with the zinc. The electrothermic distillation process presents the following advantages:

1. Large scale units mechanically operated as in the metallurgy of the other common metals.
2. Low labor, mainly unskilled.
3. Practically complete extraction of the zinc from the ore.
4. High recovery of zinc and most of it as liquid metal.
5. Low power consumption, less than electrothermic smelting and less than in the electrolytic process.
6. Applicable to all types of zinc ores with no limitations on iron, lead, lime, or other substances now barred by the retort process.
7. Applicable to complex ores, for example, such as the Burma ores.
8. Simplicity of plant and operation. Simpler than the retort process and the electrolytic process.
9. No regular consumption of fire clay; can be built in small units in out-of-the-way places where electric power is available.
10. Lower cost of operation than the retort process or electrolytic process as set forth further on.
11. May be introduced as an adjunct to the retort process for the treatment of the blue powder produced by that process. In plants using nonregenerative gas-fired Hegeler retort furnaces with waste-heat boilers, excess steam for electric power is available.

Cost of proposed electrothermic plant.

[Capacity, 2,000 to 2,500 tons of green concentrate per month. The figures are based on the flow sheet (fig. 32).]

Calcine bins, 200 tons, and elevator	\$15,000
Coke-crushing bins complete with rolls, screens, and 200-ton storage bin	20,000
Mixing system, drying, pitch melting up to the briquet process (this includes preliminary concrete mixer for mixing calcine and coke)	22,000

Three briquet presses and table conveyer.....	25,000
Baking oven and equipment and 27 charge cars.....	22,000
Nine retorts.....	20,000
Nine rams.....	30,000
Three condensers.....	10,000
Twenty-seven 200-kv. a. single-phase transformers and switchboard equipment ^a	70,000
Buildings and cranes.....	75,000
Miscellaneous.....	20,000
	329,000

^a This is based on the assumption that high-tension power can be purchased. If power is generated at the plant, an investment of \$500,000 is necessary for a 5,000-kw. plant based on the price of \$100 a kilowatt.

The cost of a roasting and sulphuric acid plant will be about \$700,000.

CALCULATION OF COST OF ELECTROTHERMIC DRY DISTILLATION PROCESS.

Basis, 1 ton of roasted concentrate, produced from 60 per cent green concentrate. Sixty tons roasted concentrate per day, or 2,000 tons green concentrate per month. Sixty tons calcine; 36 tons coke; 12 tons pitch or asphalt per day. Cost based on flow sheet. Average cost of labor at \$3 per man per shift.

I. Cost of calcine delivered to furnace in form of baked briquets
1 foot square, 27 inches long.

II. Cost of furnacing and producing metal ready for shipment.

III. Cost of material, coke, and pitch.

IV. General expense.

V. Power.

Item I. Cost of delivering briquets to furnace:

		Cost per ton calcine.	
1. Unloading 60 tons calcine, 36 tons coke, 12 tons pitch, 2 men, at \$3.....	\$6.00	\$6.00	\$0.10
2. Crushing 36 tons coke and briquets in 8 hours—			
1 man, at \$3.....	3.00		
Power.....	.50		
		3.50	.06
3. Conveying and mixing ore and coke, 8 hours—			
1 man, at \$3.....	3.00		
Power, etc.....	1.00		
		4.00	.07
4. Mixing for briquets, 24 hours—			
6 men, at \$3.....	18.00		
Power and fuel.....	24.00		
		42.00	.70
5. Briquetting, 24 hours—			
6 men, at \$3.....	18.00		
Power, etc.....	2.00		
		20.00	.33
6. Miscellaneous—			
2 men, at \$3.....	6.00		
Supplies.....	7.00		
		13.00	.22
			1.48

Item II. Cost of furnacing and producing metal for shipment:

972 briquets=27 charges, 9 retorts, 3 condensers.

1. Furnace men, 3 men, at \$6-----	18.00		
2. Helpers, 3 men, at \$3-----	9.00		
3. Metal drawers, 3 men, at \$3-----	9.00		
4. Metal to storehouse, 3 men, at \$3-----	9.00		
5. Repair gang, 3 men, at \$5-----	15.00		
		60.00	1.00

Item III. Cost of material:

1. Coke, ^a 18 tons, at \$5.50-----	99.00		
2. Pitch, 12 tons, at \$9-----	108.00		
		207.00	3.44

Item IV. General expense:

1. Superintendence-----	10.00		
2. Office expense-----	10.00		
3. Laboratory expense-----	15.00		
4. Yardmen, 3, at \$3-----	9.00		
5. Taxes and insurance and miscellaneous-----	60.00		
6. Repair supplies-----	50.00		
		154.00	2.57

Item V. Power:

1. 1,400 kw. h., at 5 cents-----	7.00	7.00	
			15.49

Calculated to the basis of raw concentrate, assuming 15 per cent loss in roasting, equals-----	13.13		
Credit on 9 per cent extra recovery, 108 pounds zinc, at 5 cents-----	5.40		
			7.73

^a Spent briquets used once again.

This does not include capital investment and amortization.

THE NATHUSIUS FURNACE.

Hans Nathusius in 1918 described several forms of a furnace in which the electrodes were embedded in the furnace lining, which was made of some material, such as dolomite, that would become an electrical conductor when hot. The heat was produced partly by the current flowing in this furnace lining, and partly by current flowing through the charge from one part of the lining to another. A uniform temperature was thus produced throughout the reaction zone.

In the furnace shown in Figure 37 the charge is fed into the pre-heating chamber *g* and thence down into the reaction chamber *r*, which is provided with electrodes embedded in the furnace lining. The zinc vapor and CO gas are drawn off through the passages *e* into the condenser *c*, where the zinc is condensed. The CO gas from the condenser is blown back through *f* into the preheater through the passages *d*, where it serves to preheat the incoming

charge. The exhaust gas is finally removed through the flue *b*. The furnace can be run on material which will remain granular during distillation, in which case the ashes are drawn off from an ash pit below the reaction zone, or if desired a fusible charge can be melted and the slag and bullion tapped from a crucible below the smelting zone.

Nathusius designed this furnace to waste as little heat as possible and in a test run reduced zinc oxide with a power consumption of 920 kw. h. per ton. He does not claim, however, that this could be duplicated in commercial work.

THE ZINC PLANT OF THE ELEKTROME-TALLURGISCHE WERKE, HORREM, GERMANY.

In 1919, A. J. Allmand and E. R. Williams described the electrothermic zinc plant of the Elektrometallurgische Werke, near Cologne, Germany.^{35c} They state that this plant began operations in September, 1913. They do not give the name of the inventor

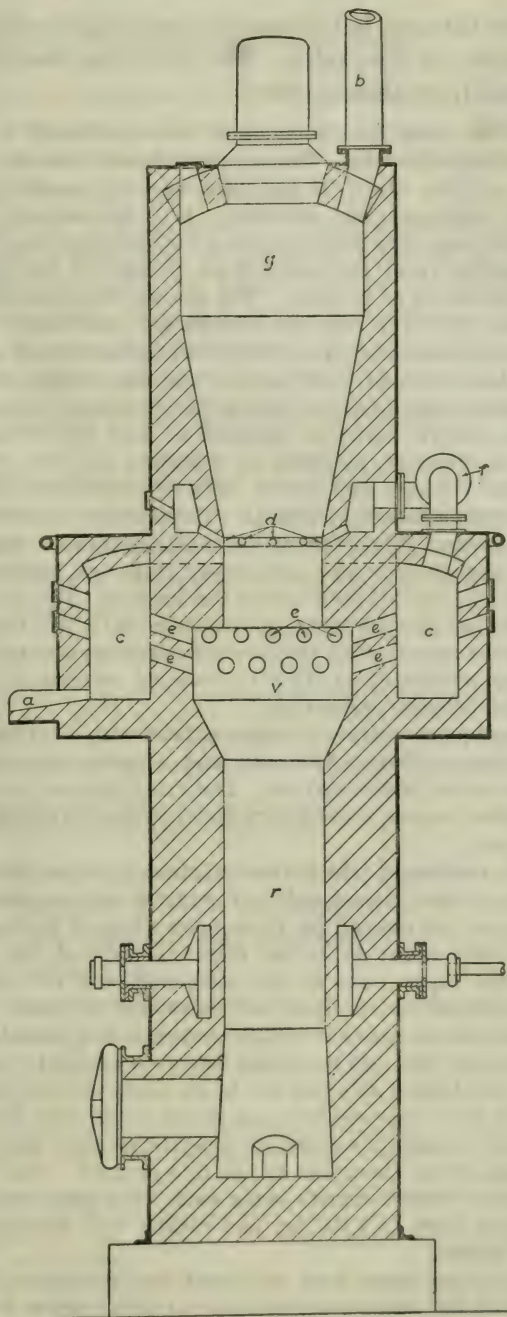


FIG. 37.—The Nathusius furnace (German Patent 293344, Aug. 9, 1916): *a*, Spout; *b*, flue; *c*, condenser; *d*, *e*, *f*, passages; *g*, preheating chamber; *r*, reaction chamber.

^{35c} Allmand, A. J., and Williams, E. R., Some chemical plants in the Cologne area, part II: Jour. Soc. Chem. Ind., vol. 38, Aug. 30, 1919, pp. 303-304 R.

of the furnace, but it seems to be similar to the furnaces designed by Queneau or Specketer. The following description of the process is quoted from their paper:

In this plant zinc is obtained electrothermally from a crude zinc oxide by reduction with coke. The ore used comes from the Harz, Ludwigshafen, etc., and contains about 70 per cent zinc with small quantities of lead, copper, silver, antimony, and cadmium. It is first mixed, slightly damp, with finely divided coke, and preheated to a dull red heat in a revolving roasting furnace (Bruchner type) by means of an oil and tar jet burner. This operation was said to take 8 to 10 hours. The charge, weighing about 5 to 5½ tons, and consisting of 70 per cent ore and 30 per cent coke, is then transferred to the electric furnace, hot from a previous operation, and smelted, this process taking anywhere from 24 to 36 hours. The zinc distills off, and is for the most part condensed and tapped as liquid, the remainder being burnt to zinc white.

The electric furnaces consisted of steel cylinders, 15 feet in length and 15 feet in diameter, mounted on trunnions and rotated on a horizontal axis by means of a gear and pinion. The shell was lined throughout by 18 inches of refractory chamotte material. Three charging doors were disposed symmetrically around the middle of the casing. Through each end of the latter passed four sets of electrodes, and copper rings and brushes made it possible to supply current in all positions of the furnace. The electrodes were of amorphous carbon rod about 5 inches in diameter and 3 feet long. From statements made it would appear that they simply carry current to the charge, which is sufficiently conducting to allow the heating currents to pass through it (high proportion of coke, preheating).

Three-phase current is employed, and with a current of 2,000 to 3,000 amperes, a voltage of 160 to 170 volts, and a power factor of 0.8; each furnace consumes about 500 to 600 kw. The exact figures, as also the time taken for the complete process, depend very much on the nature of the ore, skill of the workmen, etc.

The condenser, which abutted directly on the furnace, consisted of a brick-lined cylinder, also capable of rotation on a horizontal axis. The end of the furnace was pierced by three holes about 8 inches in diameter, placed symmetrically about 18 inches from the axis of the cylinder. To these corresponded three holes in the condenser, and the zinc vapors passed directly through and for the most part condensed to liquid. Uncondensed vapors were burned at the open end of the condenser and passed into a stack along a series of cooling flues and so to bag filters connected to a suction fan. The metallic zinc was tapped off every five to six hours and run into iron molds.

The total zinc recovery, as stated to us, was low—only 75 to 77 per cent. Of this, a certain amount is collected as zinc white (80 per cent zinc), each charge giving about 150 kg. of this product. The metal itself is 98 per cent pure, the chief impurity being lead. The slags from a single operation weigh at least 1,600 to 1,700 kg. and consist very largely of coke. At no time are they molten.

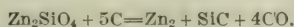
The plant when seen contained two preheaters and five electric furnaces. Four of the latter are constantly in action when working under full pressure, under which conditions a total of 300 tons of coke (for all purposes) and 10 tons of electrodes were stated to be required per month. Three hundred tons of zinc was made during the last half of 1918. Practically no work was done in November and December.

MISCELLANEOUS PROCESS AND FURNACES.

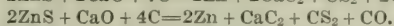
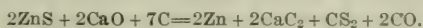
In addition to the more important furnaces described in the preceding pages, the following deserve mention either because of their historical interest or because of their possible bearing upon the future progress of electrothermic zinc metallurgy.

In 1889 an English patent³⁶ was granted to Thomas Parker on an electric smelting furnace which had the general form of a blast furnace, with electrodes introduced on either side of the smelting zone at the bottom of the shaft. The top of the furnace was closed and the charge was fed in by a screw conveyer. The following year a patent³⁷ was granted to Thomas Parker and Alfred Robinson on the application of this furnace to the reduction of roasted zinc ores with carbon.

Alfred Dorsemagen was among the first of several to attempt to smelt zinc ores in an electric furnace and to produce at the same time some valuable by-product. Dorsemagen's plan³⁸ was to reduce zinc oxide to zinc vapor and leave a residue of carborundum in the furnace, according to the equation:



Oliver W. Brown and William F. Oesterle at about the same time suggested³⁹ the possibility of producing zinc, calcium carbide, and carbon disulphide in one operation according to the reactions:



Paul Danckwardt proposed⁴⁰ to smelt mixed sulphide ores with lime, coke, and sodium sulphate in such proportions as to convert the sulphur of the ore into alkaline and alkaline earth sulphides while the zinc was liberated as vapor.

H. W. Hixon did some experimental work on a small scale in Mexico, using a buried-arc type of furnace, but had no success, due to his inability to obtain either metallic zinc or blue powder pure enough to be sold as such.⁴¹

³⁶ Parker, Thomas, Improvements in electrical furnaces and their manipulation, British Patent 17060, 1889.

³⁷ Parker, Thomas, and Robinson, Alfred, Improvements in the manufacture or production of zinc from oxidized ores of zinc, British Patent 12432, 1890.

³⁸ Dorsemagen, Alfred, Working zinc and substances containing silicic acid in electric furnace, U. S. Patent 716008, Dec. 16, 1902.

³⁹ Brown, O. W., and Oesterle, W. F., Metallurgical process, U. S. Patent 742830, Nov. 3, 1903. The electric smelting of zinc: Trans. Am. Electrochem. Soc., vol. 8, 1904, pp. 171-186.

⁴⁰ Danckwardt, Paul, Process of recovering zinc from sulphide ores, U. S. Patent 746798, Dec. 15, 1903.

⁴¹ Hixon, Hiram W., Electrothermic zinc smelting: Eng. and Min. Jour., vol. 101, June 17, 1916, pp. 1080-1081.

In 1908, Lindblad and Stalhane proposed to smelt zinc ores in a furnace similar to the Swedish electric iron-ore smelting furnace.⁴² The smelting would take place in the crucible of the furnace and any zinc vapor escaping up the shaft would be condensed in the descending charge and returned to the smelting zone, where it would progressively enrich the zinc vapor there produced, part of this being continuously withdrawn through an outlet at one side of the smelting crucible and condensed to spelter. The principle was somewhat similar to that of F. T. Synder's shaft furnace.

H. Takubo did considerable experimental work with a furnace of the buried-arc or slag resistance type.⁴³ He smelted about 1½ tons of charge per 24 hours, with a power consumption of 1,300 kw. h. per ton of charge, and a recovery of 70 per cent of the lead and zinc.

A rotatable arc furnace designed by C. A. Weeks is shown in Figure 38. Weeks has stated that he successfully produced considerable zinc at a commercial cost from zinc scrap and dross, but was not so successful in treating ores.⁴⁴

Heinrich Specketer did a great deal of work in the smelting of zinc ores, using a variety of forms of rotating furnaces,⁴⁵ and was reported to have solved the condensation difficulty.⁴⁶

F. W. Highfield designed a furnace⁴⁷ in which the raw sulphide ore was first blast roasted to expel sulphur by means of air introduced into the bottom of the furnace; after this the blast was turned off and the charge smelted in the same furnace by electrical heat.

L. G. Rowand proposed⁴⁸ to allow the charge of ore and reducer to pass down a vertical shaft whose walls were lined with current-carrying resistors to furnish the necessary heat, or down a shaft which contains staggered rows of horizontal bars to serve as resistors.

A. G. Betts has suggested treating zinc ores or zinc slags with ferrosilicon or silicon for the reduction of the zinc.⁴⁹

Henry Swift Kimball in 1914 first suggested subjecting zinc vapor in the condenser to the action of an electric field as in the Cottrell process, claiming that this gave an increased recovery of liquid spelter.⁵⁰

⁴² Lindblad, A. R., and Stalhane, O., Method and furnace for the reduction of zinc, British Patent 25979, 1909.

⁴³ Ingalls, W. R., Unpublished report to Canadian Department of Mines on the electrothermic smelting of zinc ores.

⁴⁴ Weeks, Chas. A., Milling nonferrous metals in an electric furnace: Met. and Chem. Eng., vol. 9, 1912, p. 363.

⁴⁵ Specketer, Heinrich, Process of and apparatus for producing zinc and other similar metals, U. S. Patents 1099211, June 9, 1914; 1231084, January, 1915.

⁴⁶ Ingalls, W. R., Zinc: Mineral Industry, vol. 21, 1912, p. 895.

⁴⁷ Highfield, F. W., Smelting of ores and apparatus therefor, U. S. Patent 1167176, Jan. 4, 1916.

⁴⁸ Rowand, Lewis G., Electric furnace, U. S. Patents 1289055, 1289056, Dec. 24, 1918.

⁴⁹ Betts, Anson G., Process of smelting zinc ores, U. S. Patent 918648, Apr. 20, 1909.

⁵⁰ Kimball, Henry Swift, Metallurgy, U. S. Patent 1189830, July 4, 1916.

In recent years, Sven Hultdt of Sweden has been granted several patents⁵¹ on methods of melting blue powder and refining zinc. The principle of his devices for melting blue powder is to heat the powder in a closed furnace to a temperature above the melting point of zinc, at the same time subjecting it to a rubbing action by giving the furnace a rotating or oscillating motion.

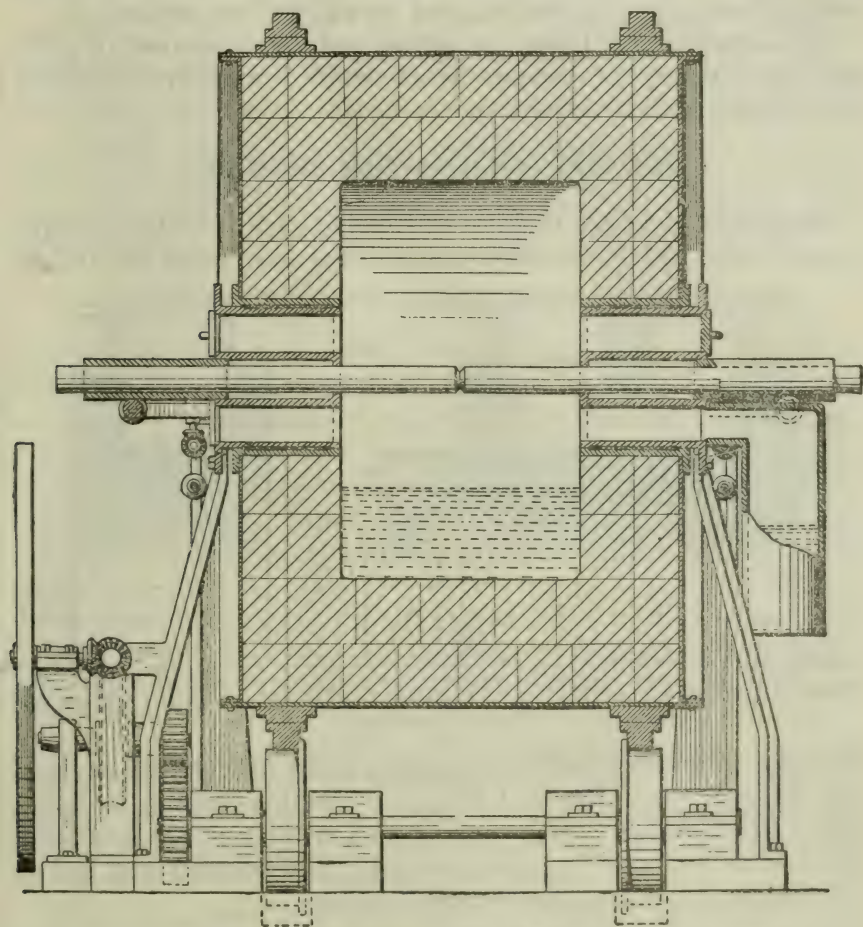


FIGURE 38.—Weeks rotatable arc furnace (U. S. Patent 949511, 1910).

Figure 39 is a diagram of E. S. Berglund's condenser for accomplishing the same result. The mixture of fluid zinc and powder collects in the bottom of the condenser and is pushed out through the opening 5 by means of the screw conveyer 3. This screw subjects the zinc powder both to a rubbing action and to a certain pressure,

⁵¹ Hultdt, Sven, Method of converting zinc powder into liquid zinc, U. S. Patent 1266808, May 21, 1918. Method of refining volatile metals, U. S. Patent 1298722, Jan. 4, 1919. Condenser for zinc vapors, U. S. Patent 1311604, July 29, 1919. Method of refining zinc, U. S. Patent 1312150, Aug. 5, 1919.

and materially increases the yield of metallic zinc. Berglund has also designed a condenser which will deliver the blue powder back to the smelting furnace as fast as it is condensed,⁵² and other improvements in the details of the operation of electric zinc furnaces.⁵³

E. A. Johansson has designed a condenser⁵⁴ in which the lower part of the condenser rotates, subjecting the blue powder to the action of stones or weights dragging on a horizontal surface.

The suggestions of Huldt, Berglund, and Johansson are all probably developments of the process as carried out at Trollhattan or other Scandinavian works.

THE ZINC CONDENSATION PROBLEM.

The greatest difficulty that has been encountered in all the attempts to smelt zinc ores electrically, and one that has caused the failure

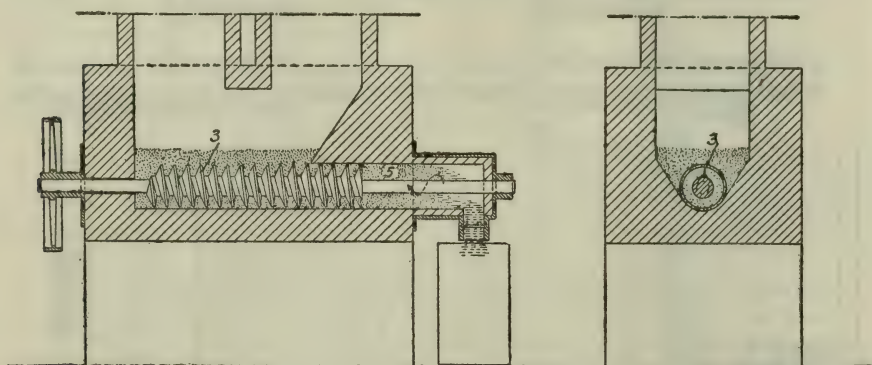


FIGURE 39.—Berglund zinc condenser (U. S. Patent 1331740, Feb. 24, 1910) : 3, Screw conveyor ; 5, opening.

of many otherwise promising attempts, is the difficulty of condensing the zinc vapor obtained from the smelting furnace to liquid zinc. In the earlier work, the major part of the zinc was invariably obtained as blue powder, which clogged the condensers and had to be resmelted to obtain a marketable product.

CONDENSATION OF PURE ZINC VAPOR.

Pure zinc vapor at atmospheric pressure begins to condense at 920° C. and will continue to condense at that temperature as long

⁵² Berglund, E. S., Method and apparatus for extracting zinc, U. S. Patent 1236395, Aug. 14, 1917.

⁵³ Berglund, E. S., Treating materials in electric furnaces, U. S. Patent 1160244, Nov. 16, 1915. Apparatus for preventing sintering in electric furnace, U. S. Patent 1196202, Aug. 29, 1916. Method of and apparatus for obtaining zinc, U. S. Patent 1207127, Dec. 5, 1916. Electrotrophic extraction of zinc, U. S. Patent 1271267, July 2, 1918.

⁵⁴ Johansson, E. A., Condenser for zinc and lead vapors, U. S. Patent 1145685, July 6, 1915.

as more vapor is supplied to keep up the pressure. As rapidly as the zinc is deposited on a condensation surface, fresh vapor will move up to take its place, and as long as the temperature is properly regulated very little or no blue powder will be produced. The temperature may vary between rather wide limits, as zinc should be condensed to liquid at any temperature between the melting point of zinc and its boiling point. In practice there has been no great difficulty in distilling crude spelter or zinc scrap and obtaining a good yield of refined spelter. The Trollhattan and Sarpsborg plants were able to treat scrap and drosses at a profit long before they had any success with zinc ores, and more recently Thomson and Fitzgerald and others have redistilled considerable amounts of crude spelter in continuous runs, with almost complete condensation to liquid zinc.

CONDENSATION OF ZINC VAPOR DILUTED WITH CO GAS.

When reducing zinc oxide, however, according to the reaction $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$, a mixture of carbon monoxide and zinc vapor is produced in equal parts by volume. In a mixture of zinc vapor with other gases, zinc starts to condense when the partial pressure of the zinc vapor in the mixture is equal to the saturated vapor pressure of molten zinc. Zinc, therefore, will not start to condense from a mixture of gases at atmospheric pressure until a temperature is reached considerably below that at which pure zinc vapor will condense; then as zinc condenses the partial pressure of its vapor in the diluting gases is decreased and the temperature must be still further lowered to condense another increment of zinc. This must be continued until the amount of zinc vapor in the uncondensed gases is reduced to the desired amount. This is analogous to the condensation of water vapor from air. If the air is very nearly saturated, a slight lowering of the temperature will cause some moisture to condense, but comparatively dry air must be cooled to a much lower temperature before moisture begins to form.

Table 5⁵⁵ gives the temperatures at which condensation starts from various mixtures of zinc vapor and carbon monoxide, also temperatures at which the condensation is 50 per cent complete and 90 per cent complete, respectively.

⁵⁵ Stansfield, Alfred, *The electric furnace*, 2d ed., 1914, p. 331.

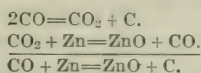
TABLE 5.—*Condensation of zinc from mixtures of zinc vapor and carbon monoxide.*

Zinc in mixture.	Condensation starts.	50 per cent condensed.	90 per cent condensed.
<i>Per cent.</i>	<i>Temp., ° C.</i>	<i>Temp., ° C.</i>	<i>Temp., ° C.</i>
100	930	930	930
90	918	910	860
80	908	890	821
70	895	872	788
60	880	853	758
50	865	832	737
40	846	810	718
30	823	783	690
20	792	750	660
10	743	705	620

Zinc does not begin to condense from the mixture of equal volumes of CO and zinc vapor resulting from the reduction of zinc oxide by carbon until the temperature falls to that at which the vapor pressure of zinc is equal to half an atmosphere. This is at about 865° C. At 600° C. $1\frac{1}{2}$ per cent of the zinc is still uncondensed. Furthermore, it requires an appreciable time for the zinc vapor to diffuse through the diluting gas to the surface of the condenser, so that if the temperature drops too suddenly the zinc will condense in the gas stream as minute drops which will be swept on and solidified as powder before they have an opportunity to coalesce. Any zinc vapor which remains uncondensed at the melting point of zinc (419° C.) will condense in the solid, powdery form as hoar frost. This latter, however, in the presence of only an equal amount of diluting carbon monoxide is less than 1 per cent of the total zinc.

In order to obtain a good condensation under the above conditions, it is necessary that the temperature of the condenser should be carefully regulated, with a gradual fall from 865° C. to only slightly above the melting point of zinc; that sufficient condensation surface be provided and so arranged that the zinc vapor need not diffuse through too great a distance in order to come in contact with it; and that the gas stream will pass through the condenser at a low enough velocity to allow ample time for the required diffusion and for small drops to coalesce and join the main body of liquid zinc before they reach the temperature at which they will solidify. As against these conditions we have the tendency of CO to dissociate at the temperature of the zinc condenser into carbon and carbon dioxide, according to the equation $2\text{CO}=\text{CO}_2+\text{C}$. This reaction is reversible. The mixture of CO_2 and CO in equilibrium with solid carbon at 1,000° C. contains only traces of CO_2 , insufficient to oxidize zinc, but as the temperature falls the CO_2 increases until at the lower temperatures of zinc condensation (500° to 750° C.) CO_2 at equilibrium is over

50 per cent. Any CO_2 in the gas mixture above a small per cent has a strong oxidizing effect on zinc, and the final result would be



or a reversal of the carbon reduction reaction, were it not for the fact that the decomposition of CO to CO_2 and C takes place very slowly in the absence of catalyzers, and under favorable conditions the zinc is condensed before the above reactions have had time to take place to a marked extent. The tendency, however, is to coat the newly formed globules of zinc with zinc oxide and prevent their coalescence. To prevent oxidation of zinc because of this formation of CO_2 and that due to infiltration of air into the condenser, it is essential that the passage from the furnace to the condenser be as short as possible and that the zinc be condensed as rapidly as it can be without too sudden cooling.

EFFECT OF IMPURITIES IN THE ZINC VAPOR OBTAINED FROM SMELTING ORES.

Complex as is the problem of condensing zinc vapor from the reduction of pure zinc oxide under the most favorable conditions, when the question of condensing the vapor from the gases produced in smelting ores is considered, a multitude of new complexities is introduced. Zinc is oxidized by the CO_2 formed from the breaking up of carbonates and from the reduction of iron or other metal oxides at low temperatures, according to reactions such as $\text{FeO}+\text{CO}=\text{Fe}+\text{CO}_2$ and by the water vapor from moisture in the ore or from hydrous minerals. The zinc vapor is diluted more extensively because CO is formed by the reduction of other metals than zinc. The zinc vapor is, moreover, sulphidized by sulphur and SO_2 , and considerable amounts of very fine dust from the charge are likely to be carried over into the condenser by the rush of evolved gases, which will hinder efficient collection of the condensed drops of zinc.

An immense amount of time and work has been put in by all investigators of the subject in trying to solve these questions in their relation to the electrothermic metallurgy of zinc; and the variety of processes, furnaces, and condensers which have been developed is largely the result of the efforts in this direction.

TWO CLASSES OF BLUE POWDER.

The problem of blue powder may be considered in its relation to the electric smelting of ores under two heads. Blue powder is classed either as chemical blue powder, which is caused by an oxide, sulphide, or similar film on the condensed zinc droplets which prevents their collecting together in one mass (under this head can

perhaps also be classed the blue powder formed when a large amount of fine dust is carried over with the zinc vapor, though the action is really not chemical; or physical blue powder, which is caused by improper physical conditions in the condenser, such as poor temperature regulation, insufficient surface for condensation, or such a large ratio of volume to surface that diffusion of zinc to the surface can not take place rapidly enough, or such high velocity of the gas stream through the zone of favorable temperature that there is insufficient time for diffusion or for proper collection of the condensed globules of zinc.

CAUSES OF CHEMICAL BLUE POWDER.

Chemical blue powder results largely from conditions in the smelting furnace, though if its formation is to be prevented leakage of air into the condenser or the presence of iron or similar catalyzer of the reaction $2\text{CO}=\text{CO}_2+\text{C}$ in the condenser must not be allowed. In the smelting of zinc ores in fuel-fired retorts the cold charge is heated up uniformly and gradually to distilling temperature, so that moisture and CO_2 are expelled and easily reducible metals are reduced before zinc starts to distill. In addition, there is always present during distillation a large excess of hot carbon to convert any CO_2 formed to CO . The charge is so small and the evolution of vapors so gradual that little dust is carried over mechanically. Some of the electric furnaces of the intermittent type—as, for example, the Fulton furnace—duplicate these conditions very closely, and it is undoubtedly for this reason that they have given very good condensation results. Conditions in electric furnaces of the continuous-smelting type are quite different from those in a fuel-fired retort. Here the temperature in the furnace is not uniform, and if the raw charge is introduced cold, water vapor and CO_2 will be produced in the cooler parts of the furnace at the same time zinc vapor is being produced in the smelting zone. In those types in which the residue is tapped as liquid slag, matte, and bullion, no great excess of carbon can be present to reduce CO_2 , and in some types the heat is so localized and the evolution of gas from the smelting zone so rapid as to cause excessive dusting of the charge. The intense heat in the localized smelting zone may, moreover, cause such impurities as iron and silicon to be reduced and volatilized, and these will coat zinc globules in the condenser and hinder their coalescence.

The production of CO_2 and water vapor in the smelting furnaces can be largely prevented by preheating the charge before it enters the electric furnace; this is one of the chief advantages of preheating. Preheating also prereduces part of the metals other than zinc and thus saves some dilution of the zinc vapor in the smelting furnace.

Several experimenters have tried to correct the lack of excess carbon in the charge by passing the vapors through a column of hot carbon after they leave the furnace, but this seems to have been of doubtful value and with proper preheating is now considered unnecessary. The localization of heat and evolution of dust can be partly overcome by heating by the resistance of the charge between electrodes rather than by the arc, and the later furnaces of the buried-arc type all do this as much as possible. Some investigators have attempted to avoid most of these difficulties and at the same time to do away with the cost of roasting by treating green sulphide ores with metallic iron. This would give zinc vapor unmixed with CO or CO₂, but in practice it is found that the vapors of the metal employed and the sulphur vapors from the unroasted sulphides have as detrimental an effect as the gases produced by the carbon reduction.

CAUSES OF PHYSICAL BLUE POWDER.

The production of physical blue powder depends almost entirely upon the design of the condenser. The condenser used in the ordinary retort process is the result of development during many years of practical experience; and though it accomplishes the desired results fairly well, existing knowledge of the factors involved is not sufficient for successfully adapting its important features to the condensation of the immensely greater amounts of zinc produced from an electric furnace.

The temperature must be properly controlled. In the retort process, this is accomplished by the heat radiated from the front of the distillation furnace. In electric smelting, it has been a common custom to provide complicated means for heating or cooling the condenser artificially. It seems certain, however, that for a constant flow of gas it should be possible so to design the condenser that the radiation would just balance the heat produced from the condensation of the zinc, and extra means for controlling the temperature should not be necessary except when starting up a cold condenser.

The size of the condenser must be greatly increased to obtain the desired capacity, but how this is to be done is not known. If all linear dimensions of the retort condenser are proportionately increased, the volume is increased but the surface is not increased in proportion. If the surface is increased in proportion to the amount of vapor to be condensed, the volume has either been increased entirely too much or a complicated and unwieldy construction has been produced. It will probably be discovered later that a simple condenser of large capacity can be constructed which will keep the

essential features of the zinc retort condenser but will alter unessential ones to suit increased size. At present there is not enough known about the physics involved to determine certainly which are the essential features and which the unessential. During recent years, condensers of large capacity have been operated with satisfactory results, but they have been the result of many practical trials and more experimentation would probably be required to adopt them to smelting furnaces of different types or of much different size.

A CRITICAL STUDY OF THE RETORT CONDENSER.

F. T. Snyder has made an interesting critical study of the retort condenser.⁵⁶ He assumes the average temperature of the gas entering the condenser to be $1,050^{\circ}\text{C}$. at the center line of the retort and of gases leaving it to be 600°C . at the center line and 450°C . at the surface of the condenser. He finds that the average rate of condensation is 0.4 gm. of zinc per square centimeter of condenser surface per hour and the average cooling effect is 0.21 cal. per square centimeter per hour. The average velocity of the gases entering the condenser is 4.67 cm. per second, and of those leaving the condenser is 7.67 cm. per second; or a mean value of 5.17 cm. per second. The average time of the gas in the condenser is 10.6 seconds. The product of the velocity of the gases in centimeters per second times the lateral equivalent diffusion distance to the condenser surface (corrected to 864°C .) is 9.9 at the retort end and 9.1 at the nose end, an average of 9.5.

Snyder believes these are the important characteristics to be kept in mind when designing a condenser for an electric furnace.

PRESENT STATUS OF THE CONDENSATION PROBLEM.

In conclusion, it may be said that several investigators have succeeded in obtaining good yields of liquid zinc in large-scale tests covering many days or weeks, so that there is now no doubt that it can be done. On the other hand, Ingalls reports that the Scandinavian metallurgists no longer attempt to produce spelter in the first smelting, but find it more advantageous to run their furnaces at a high temperature to volatilize all zinc and lead, and to collect these as blue powder, which is melted and refined in a second furnace. It was formerly thought that blue powder could not be melted economically, but in Sweden and Norway it has been found feasible to do this by subjecting the powder to a rubbing action at melting temperatures.

⁵⁶ Snyder, F. T., The condensation of zinc vapor from electric furnaces: *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, pp. 317-331.

When the physical and chemical factors involved are better understood, it will undoubtedly be found that proper condensation of zinc vapor to spelter may be fairly easily accomplished with a simple, properly designed condenser and a continuous flow of vapor.

COMMERCIAL POSSIBILITIES OF ELECTRIC ZINC SMELTING.

PRESENT COMMERCIAL DEVELOPMENT.

The electric smelting of zinc ores for the production of spelter can now be termed metallurgically feasible. The question of cost then becomes of paramount importance. Up to the present time, the plants in Norway and Sweden are the only ones that have been able to achieve commercial success. Little data on their costs are available, but the plants have been able continuously to maintain and expand their operations during more than 15 years. They are favored by having very cheap power available, and it is probable that their process would not be commercially practicable in this country. Judging from published reports, their metallurgical results are not as good as have been obtained in much of the work in this country.

BASIS FOR ESTIMATING COSTS.

Consideration of the prospects for the electrothermic process in America, then, must depend upon costs estimated from results of experimental runs with furnaces of semicommercial size. Many cost estimates have been published in the technical press, some based upon purely theoretical considerations, some on laboratory experiments, but only a few impartial ones have been based on really reliable data. The record of the development of electric smelting is full of discussions of the advantages of the electric furnace over other processes. Some of these are unprejudiced reviews of the advantages and disadvantages of each process, some are the views of overoptimistic inventors, while some claims for the electric furnace have been made which are so extravagant as to discredit the whole subject. The electric furnace undoubtedly has so many advantages that it may stand on its true merits, and its progress will be best served if those who expect great things from it will realize that there are some points which must be worked out under actual operating conditions before the maximum possibilities will be realized. Moreover, the same judgment and attention to local conditions must be employed as in adapting any other process to a particular problem.

Local conditions as to cost of power, labor, material, and character of ore supply vary so much from place to place and from year to year that detailed cost estimates are not especially valuable in a

general consideration of the subject. It will be more useful to discuss separately the different items that enter into the total cost and to compare these items with the corresponding ones in competitive processes; then from these data estimates can be made by those interested in any particular local problem by taking local conditions into consideration.

POWER.

AMOUNT OF POWER REQUIRED.

The largest single item of cost in the electrothermic process is electric power. One of the popular talking points in favor of the electric furnace as compared with the retort furnace has been that the heat was generated in the charge itself instead of having to be conducted through the retort walls, and hence was used very much more efficiently. Electric power as a source of heat is, however, much more expensive than coal, so that as close an estimate as possible must be formed of the amount which will be required in a commercial-sized furnace. The heat absorbed in the reduction of zinc oxide by carbon, according to the reaction $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$, starting with cold ZnO and C and ending with cold Zn and CO, is 55,640 cal. per mole. With cold zinc oxide and carbon at the start and with zinc vapor and CO at 1,200° C. at the finish, the heat absorbed is 102,280 cal. per mole, or 1,565 kg. cal. per kilogram of zinc = 1,419,768 kg. cal. per ton (2,000 pounds) of zinc. This is equivalent to 1,651 kw. h. per ton of zinc.

In the reduction of ores additional heat must be supplied to heat the gangue and excess carbon to smelting temperature and for other reactions which take place. Gin calculated the energy theoretically required to smelt a 50 per cent calcined calamine ore as 945,500 cal. per metric ton.⁵⁷ Richards has corrected Gin's figures and obtains 1,140,180 cal. per metric ton = 1,326 kw. h. per metric ton, or 1,203 kw. h. per ton of 2,000 pounds.⁵⁸ For a recovery of 900 pounds of zinc per ton of ore this would be 2,673 kw. h. per ton of zinc produced. Fulton gives the theoretical power consumption for 60 per cent zinc ore smelted in his briquet furnace as 1,372 kw. h. per ton of ore.⁵⁹ For 90 per cent recovery, this would be 2,541 kw. h. per ton of spelter produced.

The power required per ton of ore decreases as the percentage of zinc decreases but not in direct proportion, since more power is lost as sensible heat in the increased amount of gangue present. The power per ton of zinc produced therefore increases as the percentage of zinc in the ore decreases.

⁵⁷ Gin, Gustave, *The electrometallurgy of zinc*: Trans. Am. Electrochem. Soc., vol. 12, 1907, pp. 117-139.

⁵⁸ Richards, J. W., *Metallurgical calculations*, 2d ed., vol. 3, 1908, pp. 611-614.

⁵⁹ Fulton, C. H., *Electric furnace of large capacity for zinc ores*: Trans. Am. Inst. Min. Eng., vol. 64, 1920, pp. 188-226.

In practice, radiation losses and transmission losses must be added to the theoretical power required, but, on the other hand, part of the necessary heat may be supplied by nonelectrical means and part of the power may be recovered by burning the CO produced, and in some furnaces from the sensible heat of the products.

Snyder stated⁶⁰ that the power consumption per ton of raw ore in the experimental furnaces of the Canada Zinc Co. at Vancouver over a period of eight months approximated closely the empirical formula: Kilowatt hours per ton of raw ore = $650 + 5$ times the per cent of zinc present. Applying this to different grades of ore, he arrives at the following table:

	Kw. h. required—	
	Per ton ore.	Per ton zinc.
Pure zinc oxide—80 per cent zinc.....	1,050	1,315
Calcined calamine—50 per cent zinc.....	900	1,800
Blende—52 per cent zinc.....	910	1,750
Broken Hill—26 per cent zinc.....	780	3,000

These are lower than the theoretical power consumptions given above. In Snyder's furnace, however, the zinc was condensed in the furnace and left the furnace as liquid zinc. The CO produced was used for preheating the charge. Richards has shown⁶¹ that if the zinc and CO leave the furnace at 500° C. the figures given are theoretically possible. The power consumption given for pure zinc oxide would correspond to a furnace efficiency of 88 per cent, or if the heat absorbed in chemical reactions is subtracted and the efficiency calculated on the remainder it is 46 per cent. The power requirements in the larger plant of the Canada Zinc Co. at Nelson were not published.

Salgues reported⁶² that in a 100-kw. furnace smelting 40 to 45 per cent ore 5 kg. of zinc were produced per kilowatt day, or 1 ton of zinc per 4,355 kw. h.

Stansfield smelted an ore averaging 25 per cent zinc and 25 per cent lead in a 15-kw. furnace for 800 to 850 kw. h. per ton of ore.⁶² Stansfield made efficient use of the heat in the products of reaction, but he could not produce liquid spelter because of condensation difficulties.

The power consumption at Trollhattan during the tests made by F. W. Harboard on Broken Hill slime containing 36 per cent zinc and 24 per cent lead was 2,078 kw. h. per ton of ore.⁶³ This included 500 to 600 kw. for resmelting blue powder.

The power used by the Cote and Pierron furnace is reported in various tests as from 1,700 to 2,200 kw. h. per metric ton of ore.

⁶⁰ Snyder, F. T., *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, pp. 138-139. Discussion of paper by Gin.

⁶¹ Richards, J. W., *Metallurgical calculations*, 2d ed., vol. 3, 1908, p. 616.

⁶² Stansfield, Alfred, *The electric furnace*, 2d ed., 1914, p. 323.

⁶³ *Engineering and Mining Journal*. Zinc smelting at Trollhattan; summary of a report by F. W. Harboard: Vol. 93, 1912, pp. 314-315.

The 500-horsepower furnace being built at Epierre in 1917 was expected to require from 2,000 to 2,200 kw. per metric ton, or about 1,800 to 2,000 kw. per short ton of ore.⁶⁴ This was divided in the proportion of about 75 per cent in the smelting furnace and 25 per cent in the auxiliary condenser furnace.

In the largest run with the experimental furnace of the Canadian Department of Mines at Montreal, smelting about 700 pounds of ore in 24 hours, the power consumption was at the rate of 1,610 kw. h. per ton of ore. A determination of the power consumption on one day when the furnace at Nelson, British Columbia, was running well, in April, 1914, during which time 1.17 tons of ore were smelted, showed a consumption of 1,436 kw. h. per ton of ore. The furnace had a power factor of about 75 per cent.⁶⁵

Peterson smelted 50 per cent zinc concentrate in a furnace having a capacity of 1 ton per day with 1,100 to 1,300 kw. h. per ton of concentrates.⁶⁶

Johnson reported on short runs made with a 1-ton furnace in 1912, power consumption of 1,000 to 1,700 kw. h. per ton of preheated charge and from 3,600 kw. h. per ton of spelter on Joplin ore to 9,000 kw. h. per ton of spelter on low-grade ore.⁶⁷ For a 40-day run made in 1914 he reported 1,490 kw. h. per ton of charge.⁶⁸ The ore used in this run contained about 35 per cent zinc.

In Fulton's experiments at East St. Louis, the power consumption without preheating ranged from 1,700 to 2,300 kw. h. per ton of ore, excluding the initial runs with new furnaces and some under-distilled and overdistilled charges (see p. 63). The power consumption with preheated retorts, excluding the initial run, was from 1,237 to 1,811 kw. h. per ton of ore, the zinc distilled from the charge in every case being 99 per cent or better. With the exception of the data given for Harboard's tests and for the Cote and Pierron furnace, the power for redistilling the blue powder produced must be added to the above figures. If good condensation to liquid spelter can be obtained the treatment of blue powder will be a small expense, but if much blue powder is produced this expense may be considerable.

A conservative estimate for the total power requirements of a properly constructed furnace of 5 or 10 tons daily capacity in con-

⁶⁴ Flusin, G., Recent developments of the Cote and Pierron electric zinc-smelting process: *Met. and Chem. Eng.*, vol. 18, Jan. 1, 1918, p. 17.

⁶⁵ Ingalls, W. R., Unpublished report to Canadian Department of Mines on the electrothermic smelting of zinc ores.

⁶⁶ Peterson, P. E., The electric furnace for zinc smelting: *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 222.

⁶⁷ Johnson, W. McA., The art of electric zinc smelting: *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, pp. 197-199.

⁶⁸ Johnson, W. McA., The commercial aspect of electric zinc-lead smelting: *Trans. Canadian Min. Inst.*, vol. 17, 1914, p. 122.

tinuous operation would seem to be from 3,000 to 3,500 kw. h. per ton of spelter produced, smelting a cold charge, or from 2,500 to 3,000 kw. h. with a preheated charge. It is not unreasonable to believe that this could be lowered still further when large-scale operations have been standardized. Low-grade ores may use somewhat more power per ton of spelter produced than that given, but these will usually contain valuable by-products the recovery of which with simultaneous extraction of the zinc will give an increased margin for profit.

COST OF POWER.

The cost of power varies widely in different localities so that only very general statements can be made on this subject. An electrothermic plant, like an electrolytic plant, will usually be located near a source of cheap power.

Table 6⁶⁰ gives costs of electric power for certain cities in 1916.

Table 7⁷⁰ gives the average annual production cost of the electric utilities in the superpower zone for 1919.

TABLE 6.—*Electric power costs in 1916.*

COST PER HORSEPOWER A YEAR, 24-HOUR POWER, \$5 PER CENT LOAD FACTOR, 6 DAYS PER WEEK.

Company or municipality.	500 horse-power.	1,000 horse-power.	2,000 horse-power.
Western United Gas & Electric Co., Illinois.....	\$72.75	\$72.75	\$72.75
Munroe Electric Co., Wisconsin.....	118.80	118.80	118.80
Springfield Gas & Electric Co., Springfield, Mo.....	97.00	97.00	97.00
Cleveland Illuminating Co., Cleveland, Ohio.....	49.15	<i>H. T.</i> 36.93	<i>H. T.</i> 36.93
Edison Electric Illuminating Co., Boston, Mass.....	67.70	<i>Power.</i> 67.30	<i>Power.</i> 67.00
Pittsburgh, Pa.....	41.10	34.65	31.50
Incunman (Argentina).....	162.00	161.30	161.20
People's Incandescent Light Co., Meadville, Pa.....	71.00	71.00	71.00
Utah Power & Light Co.....	54.37	50.05	47.89
The United Illuminating Co., Bridgeport, Conn.....	95.20	95.20	95.20
The Evansville Public Service Co., Indiana.....	83.36	82.00	81.00
Erie County Electric Co.....	114.00		
United States Reclamation Service, Minidoka project, Idaho.....	46.47		
The Merchants Heat & Light Co., Indianapolis, Ind.....	58.65		
City of Leeds, England.....	41.83		
Duluth Edison Electric Co.....	113.90		
Detroit Edison Co., Detroit, Mich.....	37.45		
Detroit Electric Co., Detroit, Mich., 4,700 volts.....	32.58		
Buffalo General Electric Co. (effective Sept. 1, 1915).....	50.45	50.41	50.30
New Orleans Railway & Light Co.....	53.05	49.26	48.37
The Edison Illuminating Co., Boston, Mass.....	43.55	31.89	29.80
Indianapolis Light & Heat Co.....	58.67		
New York Edison Co.....	101.42		
Omaha Electric Light & Power Co., Omaha, Nebr.....	38.74	30.24	29.89
Texas Power & Light Co., Dallas, Tex.....	26.00	31.82	27.45
Pacific Power & Light Co., Portland, Oreg.....	75.40		
Berlin Public Service Co., Wisconsin.....	240.00		
The Springfield Gas & Electric Co.....	67.50		
Pasadena, Calif.....	133.20		
Southern California Edison Co., Los Angeles.....	46.51	46.51	46.51
Portland Railway, Light & Power Co., Portland, Oreg.....	37.50	34.46	32.94
Winnipeg.....	35.66	32.66	31.16
Calgary.....	53.20	52.71	52.47

⁶⁰ Report of the Royal Ontario Nickel Commission, 1917, p. 81.

⁷⁰ Murray, W. S., and others, A superpower system for the region between Boston and Washington: U. S. Geological Survey Prof. Paper 123, 1921, p. 49.

COST PER HORSEPOWER A YEAR, 24-HOUR POWER, 85 PER CENT LOAD FACTOR.

Company or municipality.	500, 1,000, and 2,000 horsepower.
Philadelphia D.	\$118.70
Philadelphia E.	90.20
Boston D.	76.00
New York.	116.00
Washington E.	76.00
Minneapolis.	69.80
Baltimore S.	69.80
Milwaukee Std.	63.30
Cleveland (wholesale).	52.20
Providence G.	52.20
Chicago C.	63.30
Chicago C.	57.00
Buffalo.	72.65
Cobalt, Ontario.	54.74

TABLE 7.—Annual production cost of electric utilities in the superpower zone, 1919.

Geographical division.	Cost per kilowatt hour.		Cost kilowatt year effective capacity.	
	Steam electric plants.	Hydro-electric plants.	Steam electric plants.	Hydro-electric plants.
Eastern New England.	\$0.0264	\$0.0102	\$60.00	\$31.30
Western New England.	.0297	.0109	57.50	29.50
Mohawk.	.0670	.0109	57.25	36.70
Metropolitan.	.0182	.0194	52.00	54.90
Hudson.	.0375	.0147	61.20	38.60
Anthracite.	.0186	.0161	68.80	54.70
Southern.	.0202	.0066	48.20	38.10
Superpower zone.	.02124	.0094	54.30	34.70

Twenty-four hour continuous power can be obtained at power centers in Montana, Idaho, Washington, California, Utah, and Colorado at rates varying from 0.4 to 0.7 cent per kilowatt hour. Off-peak power, which could be used advantageously in an electrothermic zinc smelter, can be obtained at somewhat lower rates, in some cases as low as 0.25 cent per kilowatt hour.

COMPARISON WITH ELECTROLYTIC PROCESS.

A fair average for power required for electrolytic zinc is 3,500 kw. h. per ton of spelter produced. This is about the maximum that would be used in the electrothermic process, and it seems that the latter must have the advantage over the electrolytic process in the item of power cost. Except where fuel is costly and power cheap, this cost will be higher than the fuel item in retort smelting, but is balanced by other advantages.

LABOR.

It is difficult to estimate with any degree of accuracy the labor required for the electrothermic process, since with the exception of

the work in Scandinavia none of the trials has ever reached the stage where economy of labor was possible. It seems certain, however, that with the large units that can be used, the labor will be much less than for the retort process and probably less than for the electrolytic process.

REDUCTION FUEL.

The cost for reduction fuel is less in electrothermic smelting than in retort smelting. Continuous smelting in an electric furnace permits the use of only slightly more than the theoretically required carbon. Fulton's process requires a large amount of coke in the briquet, but the distilled briquets can be reused as coke and finally sent to a blast furnace for the recovery of the by-product metals, where the coke contained is of value as fuel in the blast furnace.

SUPPLIES AND REPAIRS.

The large cost for retorts and condensers in retort smelting is entirely absent when the electric furnace is used. Repairs to the furnace and accessories should be no more than to the furnace and accessories of a retort plant. In the continuous slagging type of electric furnace there is an expense for electrodes, but it should be no more than in many other well-known electric furnace processes. Cote and Pierron report an electrode consumption in a small zinc furnace of 25 pounds per ton of ore smelted and in a later furnace of 11 pounds per ton of ore. Johnson reports 6 pounds per ton of ore, and Peterson estimated 9 pounds per ton. In the intermittent type of furnace there is very little expense for electrodes.

METAL RECOVERIES.

The distillation of practically all the zinc from an ore is metallurgically possible in the electric furnace. In practice, the extraction is limited by the cost of power for distilling the last trace of zinc, but the practical limit is very high. After the zinc is distilled, the losses are low since the many causes of loss, such as broken retorts, retort absorption, and diffusion of zinc vapor through the walls of the retorts, are not present when an electric furnace is used. None of the experimental electric furnace runs have been long enough for obtaining accurate data on actual final recoveries, but recoveries of 90 to 95 per cent can probably be obtained in regular operation. In addition, most of the proposed electrothermic processes either recover lead, copper, and the precious metals as bullion and matte or as a product in very desirable shape for blast-furnace treatment, whereas the product of a zinc retort is very undesirable raw material for a blast furnace, as is also the usual sludge residue from an

electrolytic plant. The treatment of low-grade and complex ores in the electric furnace offers no difficulties, and the presence of the impurities which cause so much trouble in retort smelting is not detrimental. Electric smelting has the additional advantage that since zinc sulphide is decomposed by iron reduced from iron oxides in the charge a dead roast is not necessary. This decomposition of ZnS by Fe can not be utilized in the retort process because FeS formed would quickly destroy the retorts.

MANAGEMENT AND GENERAL EXPENSE.

The charge for management and general expense for a new electrothermic smelting plant, at the present stage of development of the process, would undoubtedly be greater than for a retort or electrolytic plant, for a good many operating details would have to be worked out before operations could be reduced to a routine basis; but once the process is standardized on a commercial basis the cost of management should be no greater than for a retort or electrolytic plant.

FIRST COST OF PLANT.

The cost of an electrothermic zinc plant will be less than for either the retort or electrolytic process, if power is purchased. If a power plant must be built, the total cost will be more than for the retort smelter but less than for an electrolytic plant with its power plant.

The electrothermic process has the additional advantage over the electrolytic process that a small electric smelting plant of one or more 10-ton units will be practicable, whereas the electrolytic process has given evidence that it is only economical where a large tonnage is to be treated.

SUMMARY.

Conditions being equal, the electrothermic process may be said to have the advantage over the electrolytic process in the items of power, labor, metal recovery, cost of roasting, first cost of plant, and in its adaptability to smaller scale operations; and over the retort process in the items of reduction fuel, labor, metal recovery, ability to treat low-grade and impure ores, cost of roasting, first cost of plant, and cost of retorts and condensers.

The electrothermic smelter must be near cheap power, as must also the electrolytic, but as many retort smelters are now some distance from their supply of ore this is no great difficulty. Each of the three processes—retort, electrolytic, and electrothermic—has its particular field and there are undoubtedly places in this country where the electrothermic process could be profitably applied.

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